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S. W. STRATTON, DIRECTOR

No. 184 – 202

FIRE TESTS OF BUILDING COLUMNS

BY

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Associated Factory Mutual Fire Insurance Companies

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PREFACE

The investigation herein reported was conducted in cooperation with the Associated Factory Mutual Fire Insurance Companies and the National Board of Fire Underwriters, the tests being made at Underwriters' Laboratories, Chicago, Ill.

When the assistance of this Bureau on certain portions of the work was sought during the early part of 1914, the preparatory work had been in progress for about four years, a building adapted for the tests had been erected, and the design of equipment and test columns nearly completed.

As the safety of building supports is a matter of general public interest, participation in the proposed investigation appeared justified, and a basis of cooperation was arranged whereby the work was made a part of the investigation on the fire-resisting properties of building materials that was then being initiated by this Bureau.

By the method of presentation that follows, Section I gives a general outline of the tests; Sections II, III, and IV give descriptions of the columns, column coverings, and methods used in their preparation; Section V and Appendix D, results of auxiliary tests of materials; Sections VI, VII, VIII, and IX, descriptions of apparatus and method of testing; Sections X and XI and Appendices A, B, C, results of tests; Sections XII and XIII, discussion of test data and conclusions.

The full presentation of the results of the present investigation is necessarily of considerable extent, and familiarity with the general plan will be of material aid in reading and, if desired, in selecting for particular attention such portions as may best serve the reader's immediate purpose.

FIRE TESTS OF BUILDING COLUMNS

By S. H. Ingberg, H. K. Griffin, W. C. Robinson, and R. E. Wilson

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I. INTRODUCTION

1. PURPOSE

The purpose of this investigation is to ascertain (1) the ultimate resistance against fire, of protected and unprotected columns as used in the interior of buildings; (2) their resistance against impact and sudden cooling from hose streams when in a highly heated condition.

While columns form the most important element in the strength of a building, few representative tests have been made to determine their ability to support load when exposed to fire, and fire experience has only a limited value due to the many unknown variables involved. As a consequence, wide differences in requirements relating to the protection of columns against fire exist between different municipal codes and other published regulations. This investigation was undertaken to obtain information on which proper requirements for the more general types of columns and protective coverings can be based.

2. SCOPE OF TESTS

The present series consists of 106 tests of columns, of which 91 were fire tests and 15 fire and water tests.

The fire test series includes (1) tests of representative types of unprotected structural steel, cast iron, concrete-filled pipe, and timber columns; (2) tests wherein the metal was partly protected by filling the reentrant portions or interior of columns with concrete; (3) tests wherein the load-carrying elements of the columns were protected by 2-inch or 4-inch thicknesses of concrete, hollow clay tile, clay brick, gypsum block, and also single or double layer of metal-lath and plaster; (4) reinforced-concrete columns with 2-inch integral concrete protection.

The covering materials for each class of protection were obtained from the main producing regions of the country, the object being to include samples from the principal mineralogical subdivisions that find general application in building construction. A large number of auxiliary tests of constituent materials were made, including several hundred compression tests of the concrete employed.

The test columns were designed for a working load of approximately 100 000 pounds, as calculated according to accepted formulas, the amount varying somewhat for the different sections.

The load was maintained constant on the column during the test, the efficiency of the column or its covering being determined by the length of time it withstood the combined load and fire exposure.

The latter was produced by placing the column in the chamber of a gas-fired furnace whose temperature rise was regulated to conform with a predetermined time-temperature relation. Measurements were taken of the temperature of the furnace and test column and of the deformation of the latter due to the load and heat.

In the fire and water tests the column was loaded and exposed to fire for a predetermined period, at the end of which the furnace doors were opened and a hose stream applied to the heated column, the duration of the application and the pressure at the nozzle varying with the length of time the corresponding type of column withstood the regular fire tests.

3. ACKNOWLEDGMENTS

The tests were jointly conducted by the Associated Factory Mutual Fire Insurance Companies, the National Board of Fire Underwriters, and the Bureau of Standards. The fire tests were made at Underwriters' Laboratories, Chicago, Ill. The furnace and related equipment were designed and constructed by Underwriters' Laboratories during the period 1912 to 1917, the work being coordinated with a general building program that was carried out by them at that time. Its use, except for repairs and replacements, was donated for the tests. Preliminary work relative to the testing schedule was begun as early as 1910 by the Associated Factory Mutual Fire Insurance Companies and Underwriters' Laboratories, the cooperation of this Bureau beginning in 1914. A final schedule of tests was adopted in March, 1916. The preparatory work of covering and placing the test columns extended from May, 1916, to May, 1917. With the completion of the equipment, testing of columns began in June, 1917, and was completed in December, 1918. This paper in its final form was approved by the cooperating parties in December, 1920.

The apparatus for measurng temperature and deformation was supplied by this Bureau. Auxiliary physical and chemical tests of the materials of the test columns and their protective coverings were made at the Washington and Pittsburgh laboratories of the Bureau.

The expense for labor and material was shared on an equal basis by the three cooperating units. The materials were in part purchased under the usual specifications, and in part supplied gratis by the following companies, whose cordial cooperation is herewith acknowledged:

American Bridge Co., Chicago, Ill., steel test columns.
Bethlehem Steel Co., South Bethlehem, Pa., steel test columns.
Chicago Bridge & Iron Works, Chicago, Ill., steel test columns.
R. D. Cole Manufacturing Co., Newnan, Ga., steel test columns.
Pittsburgh-Des Moines Steel Co., Pittsburgh, Pa., steel test columns.
U. S. Cast Iron Pipe & Foundry Co., Burlington, N. J., cast-iron test columns.
Lally Column Co., Cambridge, Mass., steel-pipe test columns.
National Lumber Mfrs. Assn., Chicago, Ill., timber test columns.
Southern Pine Assn., New Orleans, La., timber test columns.
West Coast Lumbermen's Assn., Seattle, Wash., timber test columns.
Corrugated Bar Co., Buffalo, N. Y., reinforcing steel.
Associated Metal Lath Mfrs. Chicago, Ill., expanded metal lath.
Zander-Reum Co., Chicago, Ill., woven-wire lath.
Chicago Portland Cement Co., Chicago, Ill., Portland cement.
Marblehead Lime Co., Chicago, Ill., lime.
Pelee Island Sand & Gravel Co., Cleveland, Ohio, sand.
Phoenix Sand & Gravel Co., New York, N. Y., sand.
American Sand & Gravel Co., Chicago, Ill., sand and gravel.
Chicago Gravel Co., Chicago, Ill., sand and gravel.
Union Sand & Material Co., St. Louis, Mo., sand and gravel.
Haverstraw Crushed Stone Co., New York, N. Y., crushed stone.
Ohio Quarries Co., Cleveland, Ohio, crushed stone.
Rockport Granite Co., Rockport, Mass., crushed stone.
White Fireproof Construction Co., New York, N. Y., hard coal cinders.
Camp Conduit Co., Cleveland, Ohio, hollow clay tile.
National Fireproofing Co., Pittsburgh, Pa., hollow clay tile.
Whitacre Fireproofing Co., Waynesburg, Ohio, hollow clay tile.
Gypsum Fireproofing Co., Chicago, Ill., gypsum block.
Keystone Fireproofing Co., New York, N. Y., gypsum block.
Bestwall Manufacturing Co., Chicago, Ill., gypsum wall board.

The participation of the Associated Factory Mutual Fire Insurance Companies was under the direction of H. O. Lacount, assistant secretary and engineer, and they were represented on the preliminary conference and preparatory work by C. W. Mowry, and at different times in conducting the tests and preparing the results for publication, by R. E. Manning, W. G. Lawrence, and R. E. Wilson.

The National Board of Fire Underwriters participated through Underwriters' Laboratories, the work being under the direction of W. C. Robinson, vice president, assisted by F. W. Frederick, W. G. Howell, and F. Taylor in designing and constructing the testing furnace and related equipment, and at different times by G. W. Riddle and R. K. Porter in conducting the tests.

The cooperation of this Bureau was under the administrative direction of C. W. Waidner, chief of the division of heat and thermometry. S. H. Ingberg and H. K. Griffin were actively associated with the work for the periods 1914 to 1920 and 1917 to 1920, respectively, the former being in direct charge of the experimental program.

Technical assistance for extended periods was given by A. J. Steiner and R. F. Zeunnert.

Mineralogical analyses of concrete aggregates were made by R. S. Knappen of the department of geology of the University of Chicago.

Acknowledgment is also due to a number of engineers, contractors, architects, and public officers who kindly examined a preliminary draft of the schedule of tests which was submitted to them. Their criticisms and suggestions were duly considered in formulating the final program.

II. DESCRIPTION OF COLUMNS

1. STRUCTURAL STEEL COLUMNS

(*a*) DETAILS OF DESIGN

In Figs. 1 to 4 are shown details of all structural sections employed in the tests. The lower 12 feet 8 inches constitute the test column proper, the upper enlarged extension, 3 feet in length, serving merely as a means for transmitting the load to the column. This head is protected by concrete as shown in Fig. 9, and, being above the ceiling line, is not directly exposed to the heat of the testing chamber.

The bottom base angles are designed to develop about one-half of the transverse strength of the column considered as a beam at ordinary temperature, and during the test are embedded in the fireproofing for the base plates of the testing machine. The top anchorage is designed to develop the full transverse strength of the column.

The test columns are provided with brackets near the top to introduce conditions affecting the application of the protective coverings similar to those obtaining in building construction.

On account of the prevailing use of the solid rolled and built-up H sections, more than one-half of the total number of steel columns used in the tests were of these types.

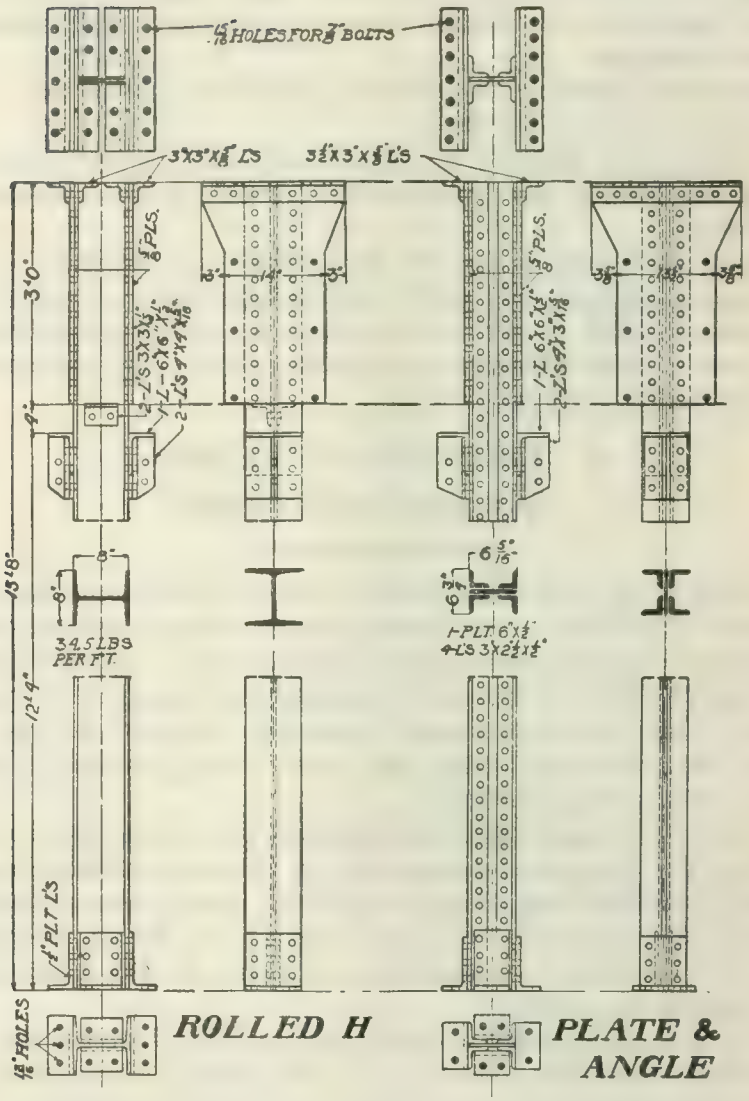


FIG. 1.—*Details of structural-steel columns. Rolled H and plate and angle sections*

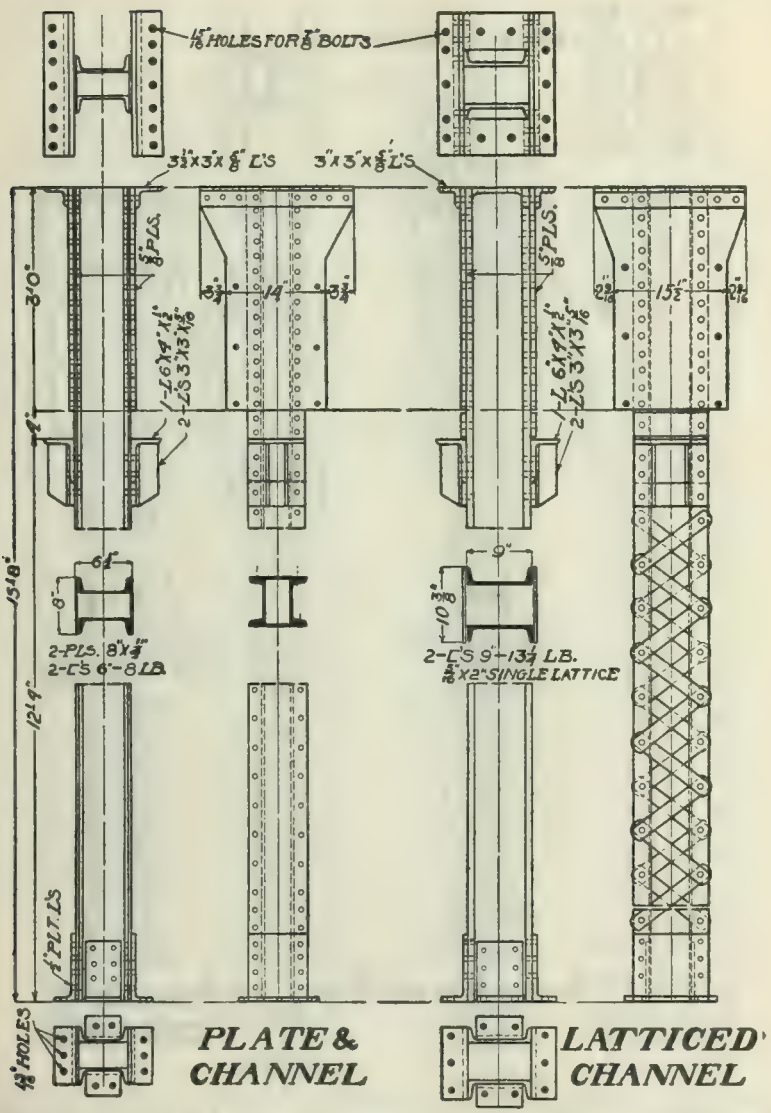


FIG. 2.—*Details of structural-steel columns. Plate and channel and latticed channel sections*

20184°—21——2

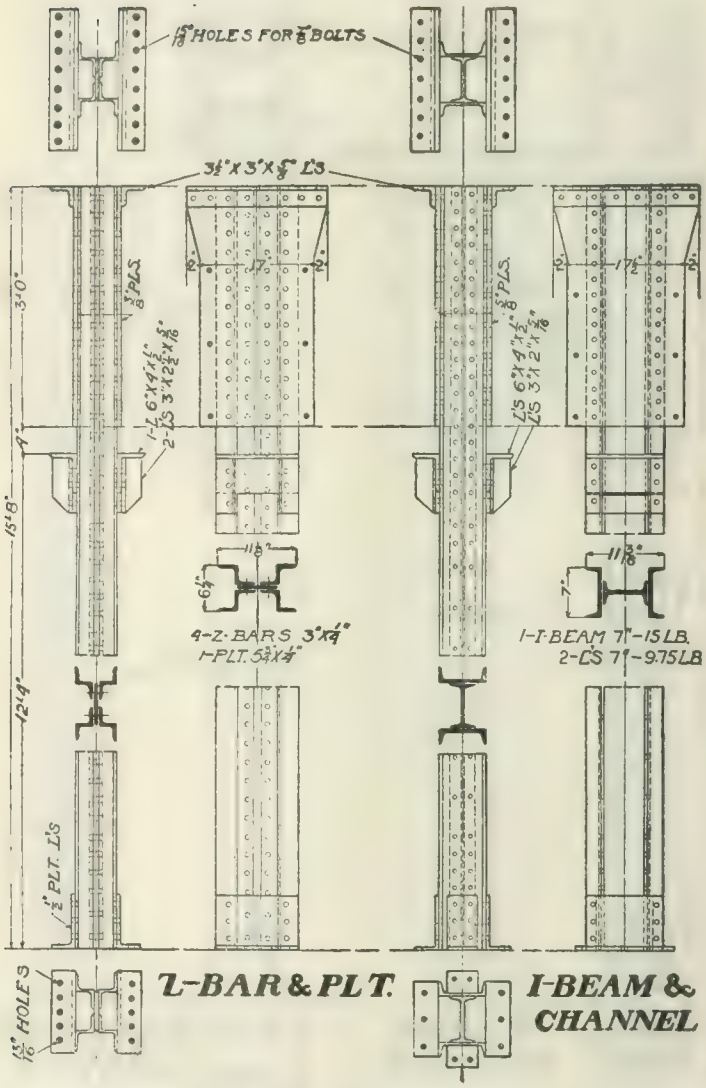


FIG. 3.—*Details of structural-steel columns. Z-bar and plate and I-beam and channel sections*

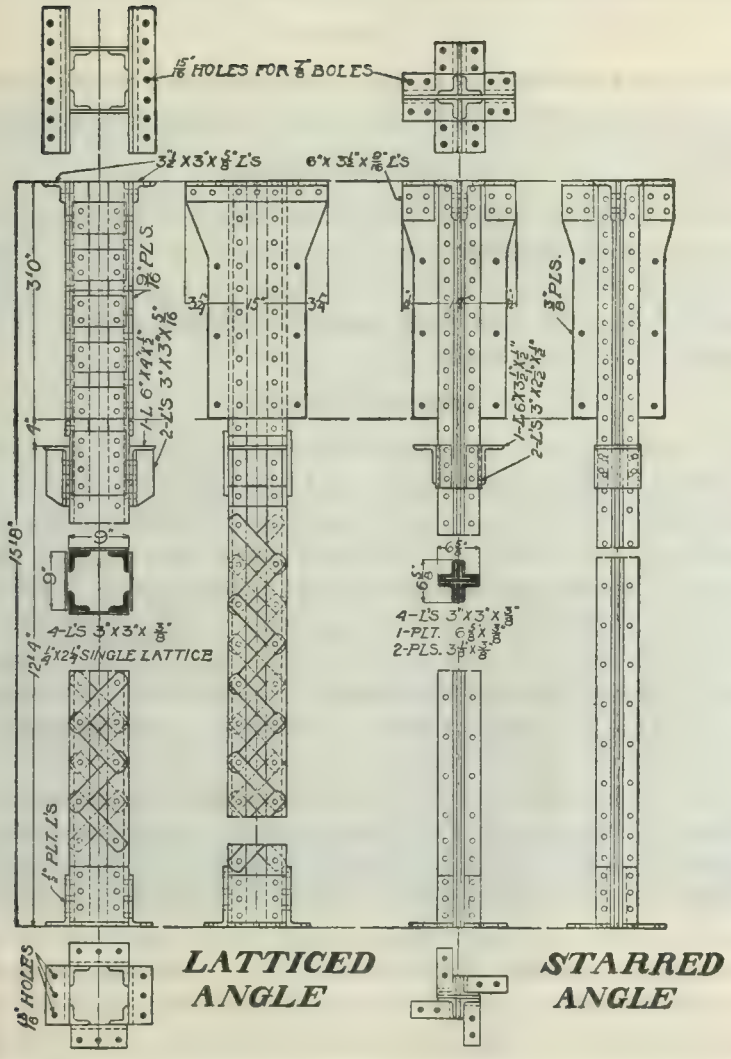


FIG. 4.—*Details of structural-steel columns. Latticed angle and starred angle sections*

(*b*) BEARINGS

The column bases were specified to be finished by milling. While this had been done in almost all cases, it was necessary to improve most of the bearings by grinding, in order not to induce uneven stress distribution in the columns when loaded.

(*c*) RIVETING

This was examined by striking the rivet heads with a hammer. Very few loose rivets were found, and these were redriven before testing.

(*d*) INITIAL STRAIGHTNESS

Before being covered or tested, the columns were examined for straightness by measuring from points on the column as placed in vertical position to a fine wire stretched from base angles to bracket. The columns were generally straight within $\frac{1}{8}$ of an inch and in all cases within $\frac{3}{16}$ of an inch.

(*e*) PROPERTIES OF SECTIONS

The area of one or more sections of each type was obtained by calipering. The special gages used are shown in Fig. 5. The large gage (*a*), having a range from 0 to 12 inches, was used for outside measurements, the distance between the point of the pin and the point of the dial stem at zero reading being obtained by means of calibrated end measuring rods. The inside caliper (*b*) has a range from $3\frac{1}{2}$ to $6\frac{1}{2}$ inches, the length between its points at zero reading being determined by measurement in the outside gage. Shape and area of fillets and corners were obtained from plaster impressions.

A comparison of nominal or handbook areas and measured areas is given in Table 1, the two being generally in agreement within 1 per cent and in all cases within 4 per cent. In calculating the loads to be carried during the fire test, the nominal area was used for all columns, the main dimensions of the section members being measured to ascertain their nominal size. The latter was found to be as called for on the details, except in the case of two plate-and-angle and two plate-and-channel columns, where the plates were $\frac{1}{32}$ of an inch heavier than required.

Values of other essential properties of the sections are given in Table 3a (p. 33).

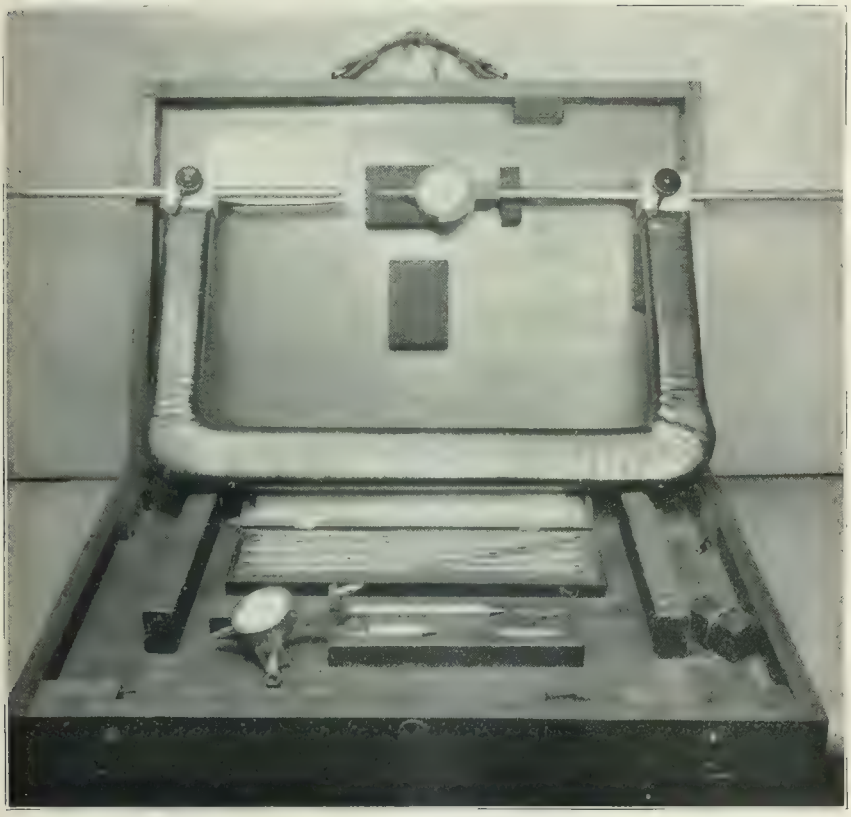


FIG. 5.—*Calipers for measuring steel shapes*

TABLE 1.—NOMINAL AND MEASURED AREAS OF STRUCTURAL STEEL SECTIONS

Test No.	SECTION	SECTION MEMBERS	Nominal Area, Sq. In.		Measured Area, Sq. In.	
15	Rolled H	Solid Rolled H 34.5 lb		10 17		10.06
30	Rolled H	Solid Rolled H 34.5 lb		10.17		10.05
50	Plate and Angle	1 Plate ½ by 6 in	3.00		2.99	
		4 Angles 3 by 2½ by ½ in	10.00	13.00	9.97	12.96
51	Plate and Angle	1 Plate ½ by 6 in	3.00		2.92	
		4 Angles 3 by 2½ by ½ in	10.00	13.00	9.94	12.86
110	Plate and Angle	1 Plate ½ by 6 in	3.00		2.99	
		4 Angles 3 by 2½ by ½ in	10.00	13.00	10.21	13.20
39	Plate and Channel	2 Plates $\frac{5}{16}$ by 8 in	4.50		4.51	
		2 Channels 6 in.—8 lb	4.76	9.26	4.73	9.24
107	Plate and Channel	2 Plates ¼ by 8 in	4.00		4.19	
		2 Channels 6 in.—8 lb	4.76	8.76	4.57	8.76
54	Latticed Channel	2 Channels 9 in.—13¼ lb		7.78		7.80
41	Z-bar and Plate	1 Plate ¼ by 5¾ in	1.44		1.46	
		4 Z-bars 3 by ¼ in	7.88	9.32	7.85	9.31
56	Z-bar and Plate	1 Plate ¼ by 5¾ in	1.44		1.48	
		4 Z-bars 3 by ¼ in	7.88	9.32	7.59	9.07
44	I-beam and Channel	1 I-beam 7 in.—15 lb	4.42		4 54	
		2 Channels 7 in.—9¾ lb	5.70	10.12	5.71	10.25
58	I-beam and Channel	1 I-beam 7 in.—15 lb	4.42		4.59	
		2 Channels 7 in.—9¾ lb	5.70	10.12	5 76	10.35
46	Latticed Angle	4 Angles 3 by 3 by $\frac{5}{16}$ in		8.44		8.45

2. CAST-IRON COLUMNS

(*a*) DETAILS OF DESIGN AND MANUFACTURE

Structural details are given in Fig. 6. The columns shown in (*a*) and (*b*) were made by a Chicago foundry experienced in the making of building castings. They were cast horizontally, with continuous core supported by chaplets, and single gate and riser, the gate being at one end and the riser at the other.

Five of these columns were tested with ends restrained by bolting to base plates at top and bottom as shown in (*a*), and two were tested with ends not restrained, as shown in (*b*). In the latter case the bolts at the bottom were omitted, and at the top the

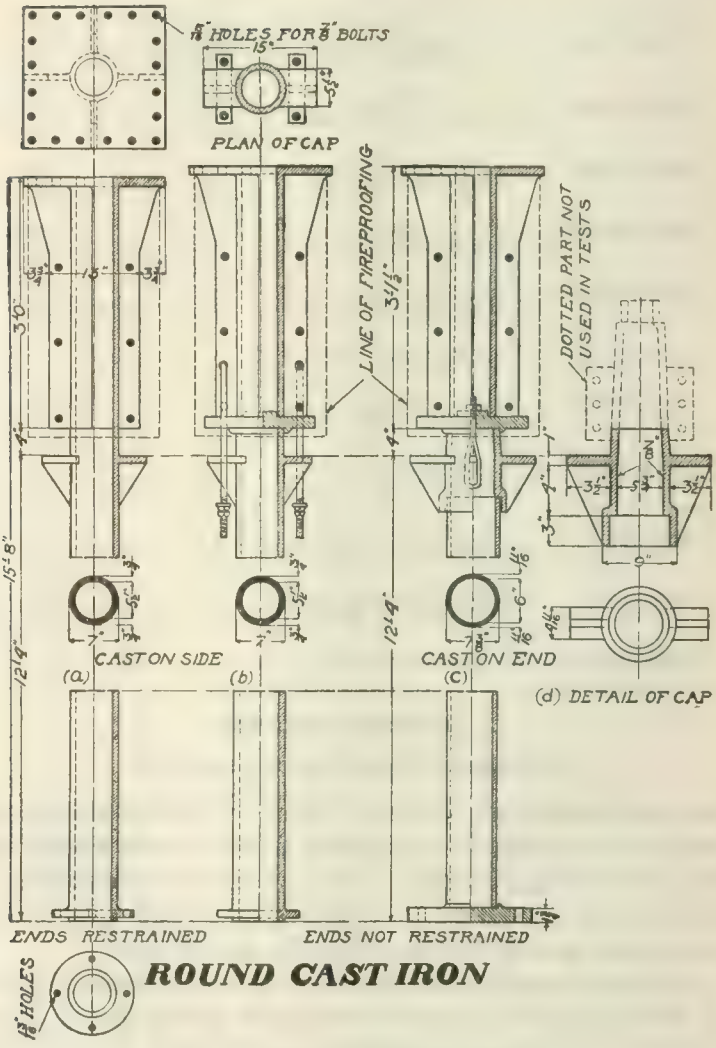


FIG. 6.—*Details of cast-iron columns*

column was cut at the junction with the head, a bearing plate being inserted between the cut surfaces. A U bolt and strap on each side served to hold the column end in case the column should break at failure.

Three columns were of the cast-iron-pipe type with detached cap, as shown in (*c*) and (*d*) of Fig. 6. These columns were cast in vertical position. In testing, substantially the same bearing details were used as for the horizontally-cast columns that were tested with ends not restrained.

(*b*) BEARINGS

The ends of the columns were milled, the bearings being in all cases fairly even and true. All bearing surfaces of caps and bearing plates were machined. The top and bottom bearings were protected by fireproofing, the same as for the structural-steel columns, the length exposed to the fire being 12 feet.

(*c*) INITIAL STRAIGHTNESS

The amount and direction of initial curvature was determined in the same manner as for the structural-steel columns. The columns that were cast horizontally were straight within $\frac{1}{8}$ of an inch, and those cast on end were straight within $\frac{1}{16}$ of an inch.

(*d*) VARIATIONS IN METAL THICKNESS

In the case of the horizontally cast columns, the core at the midpoint of the length was found to have been displaced by amounts varying from $\frac{1}{16}$ to $\frac{1}{4}$ of an inch for the individual columns. The minimum thickness of metal was $\frac{1}{2}$ an inch against a nominal thickness of $\frac{3}{4}$ of an inch. The area of metal exceeded the nominal area by amounts up to about 1 square inch. The exact effective area was difficult to determine on account of the roughness of the interior surface.

For the vertically cast columns the thickness varied from $\frac{5}{8}$ to $\frac{3}{4}$ of an inch against a nominal thickness of $\frac{11}{16}$ of an inch, the area being generally about $\frac{1}{2}$ square inch in excess of the nominal.

3. PIPE COLUMNS

(*a*) DETAILS OF DESIGN AND MANUFACTURE

Two concrete-filled pipe columns were tested, one made with a standard 7-inch pipe, and the other with a standard 8-inch pipe reinforced in the fill with four $3\frac{1}{2}$ by $3\frac{1}{2}$ by $\frac{3}{8}$ inch angles riveted back to back. Details of columns and bearings are given in Fig. 7. The details at the top are similar to those of the

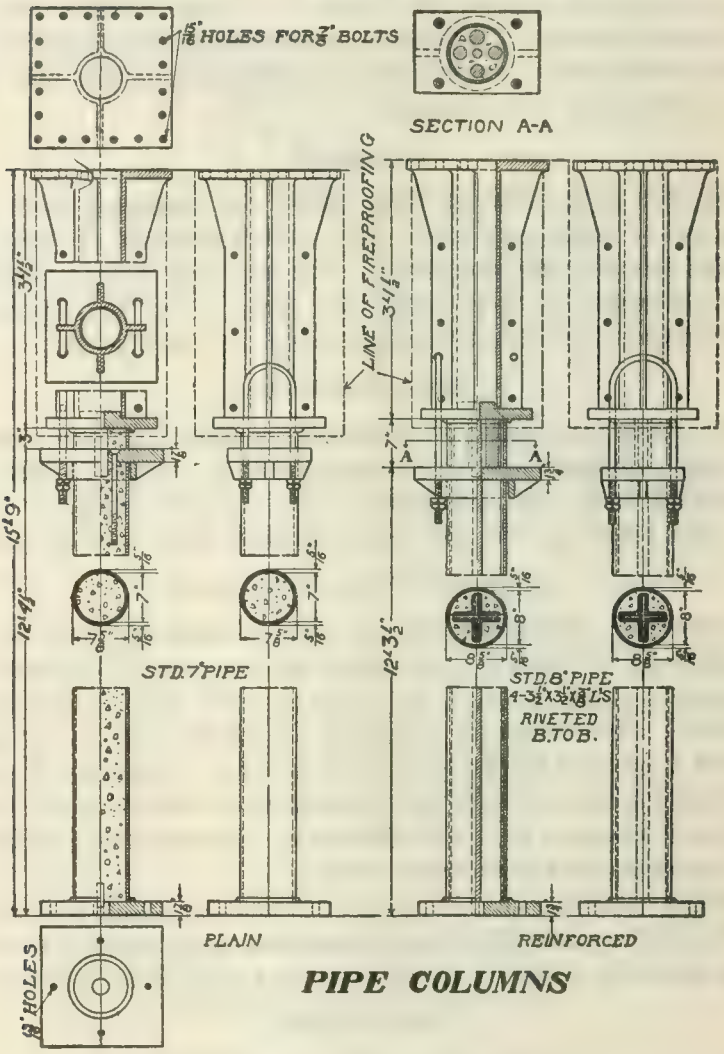


FIG. 7.—*Details of pipe columns*

cast-iron columns that were tested with unrestrained ends except that a short strut was placed between the top of the cap and the upper bearing plate to obtain conditions more nearly representative of use in buildings.

The 7-inch pipe column was filled at the manufacturer's plant especially for the test and specimens of concrete and concrete aggregates were secured. The concrete mixture was 1 part Portland cement, 1½ parts Cambridge (Mass.) bank sand, and 3 parts crushed blue trap rock quarried at Westfield, Mass. The reinforced-pipe column was obtained from a stock of completed columns, and specimens of the concrete aggregates could not be obtained, but they were said to be the same as for the 7-inch column.

(*b*) BEARINGS

The bearings were square and unrestrained. The bearing surfaces of the caps were unfinished; base and top bearing plates were machined. The pipes had sawed ends that were fairly even. The concrete on the bottom bearing of the 7-inch pipe column projected about $\frac{1}{16}$ of an inch below the pipe.

4. REINFORCED CONCRETE COLUMNS

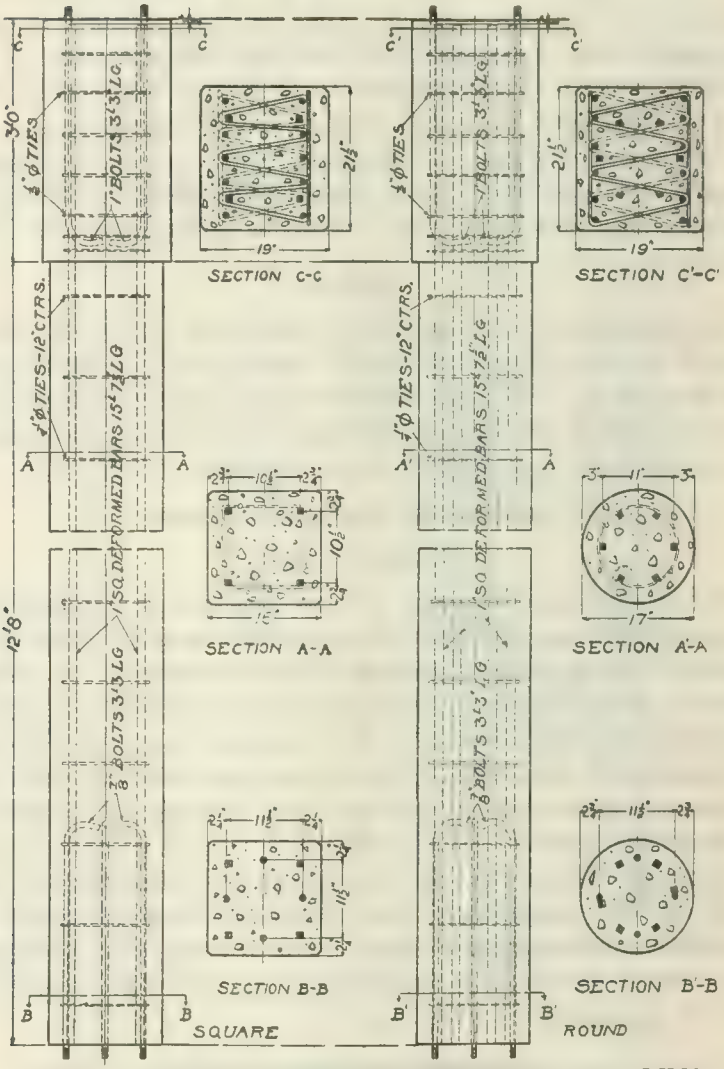
(*a*) DETAILS OF DESIGN

The types tested include square and round longitudinally reinforced columns, and round columns with lateral reinforcement of spiral hooping and longitudinal bar reinforcement. Details are shown in Figs. 8 and 9. The spiral reinforcement constitutes in volume about 1 per cent of the contained concrete. The size and spacing of the lateral ties in the longitudinally reinforced columns represent current practice with respect to this detail.

The concrete was of 1 : 2 : 4 mixture. In the columns for the fire tests, Fox River (Ill.) sand with Chicago limestone and Long Island (N. Y.) sand with New York trap rock were the two combinations of aggregates used. For the fire and water tests each column was cast in three 4-foot sections with concrete of different aggregates in each section.

(*b*) BEARINGS AND END RESTRAINT

The lower ends of the vertical bars were ground true and abutted on the base plate on which the column was cast, the upper ends of the bars terminating ½ an inch below the top bearing plate. The head was cast monolithic with the test column proper, and was



VERTICALLY REINFORCED CONCRETE COLUMNS

FIG. 8.—*Details of vertically reinforced concrete columns*

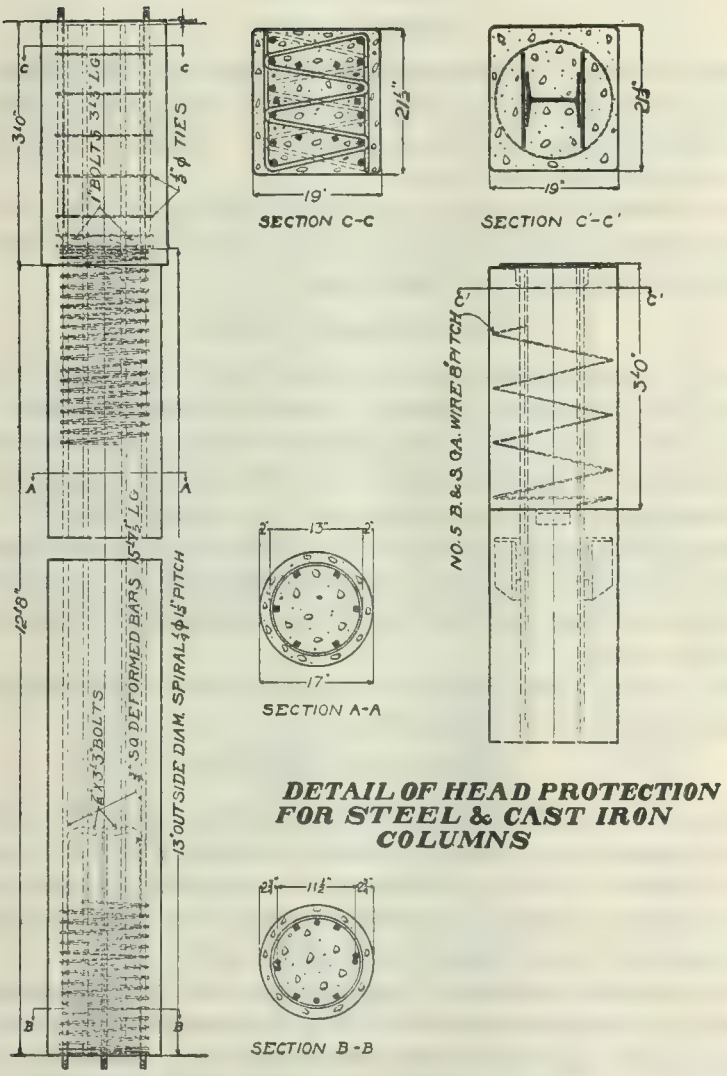


FIG. 9.—*Details of hooped concrete column and column-head protection*

suitably reinforced and anchored into the top bearing plate. The latter was set in Portland cement mortar. At the bottom, the the column was tied to the base plate with four ⅞-inch bolts.

5. TIMBER COLUMNS

(*a*) SPECIES OF TIMBER

Tests of timber columns include four with long-leaf yellow pine and two with Douglas fir, these two species being chosen on account of wide use in heavy timber construction. Details of columns and bearings are given in Fig. 10.

The yellow-pine columns were cut from timber grown in Pike County, Miss. The Douglas-fir columns came from the northern Douglas-fir region of the Pacific coast.

(*b*) CAP AND BEARING DETAILS

In the construction shown at (*a*) in Fig. 10, the load is transmitted to the column through a cast-iron pintle and cap. The side of the pintle and the top bearing of the cap are covered by timbers and flooring that frame into them in a manner similar to that used in standard applications of mill construction.

In the construction shown at (*b*) the load is transmitted through a timber strut and steel-plate cap, this method of column and beam support typifying another form of standard practice with regard to these details.

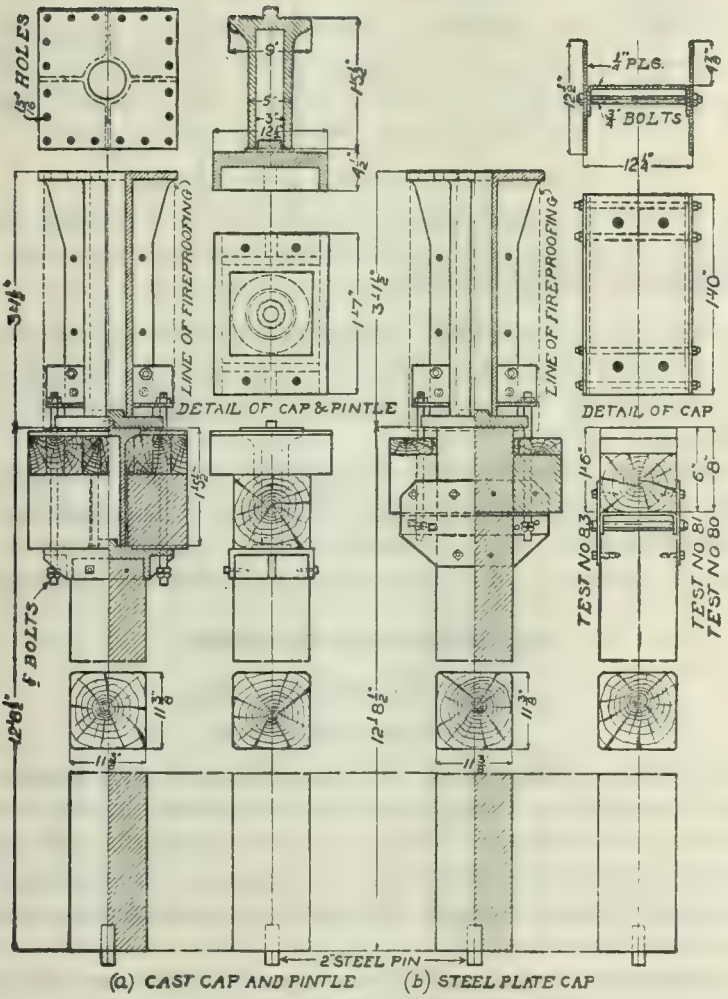
The bearings were finished so as to be fairly even and perpendicular to the axis of the column.

(*c*) PROPERTIES OF THE TIMBER

The chief characteristics of the timber are given in Table 2. The columns were select structural material with few knots or other defects. They were surfaced on the sides and the corners were slightly beveled. A nominal section of 11⅜ by 11⅜ inches was assumed in calculating the working load.

The number of annual rings per inch and the percentage of summer wood were determined on a representative line over the third, fourth, and fifth inches from the pith, the values given being the average for the two end faces. The resin content was obtained by extracting borings from representative points in the section with benzole and drying to constant weight at 70° C.

The moisture content and dry weight were determined from disks 1 inch thick cut 2 feet and 4 feet from the end of a timber of the same size as the test columns and which had been subjected to



TIMBER COLUMNS

FIG. 10.—*Details of timber columns*

TABLE 2.—PROPERTIES OF TIMBER IN TEST COLUMNS

Test No.	Species	Dimension of Section, In.	Number of Rings per In.	Summer-wood, Percent	*Rosin Content, Percent	*Moisture Content, Percent	Weight per Cu. Ft., Pounds	
							**As Tested	*Oven Dry
78	Longleaf pine.....	11 5/16 by 11 1/2	14	35			41.3	
79	Longleaf pine.....	11 1/4 by 11 1/4	17	50			.....	
80	Longleaf pine.....	11 1/4 by 11 5/16	11	35	7.07	17.1	41.3	34.7
81	Longleaf pine.....	11 3/8 by 11 3/8	14	35			46.7	
82	Douglas fir.....	11 3/8 by 11 3/8	[illegible]	33			36.1	
83	Douglas fir.....	11 3/8 by 11 3/8	10	33	1.34	26.2	39.0	30.1

*Determined from representative samples.
**Determined by weighing the test columns.

the same storage conditions. These were dried to constant weight at 100° C, the percentage moisture, which includes besides water other substances volatile at the given temperature, being based on the dry weight.

In point of general quality of material, the test columns conformed with the requirements of published specifications for structural timber. The dimensions of the finished section were smaller by about 1/8 of an inch than as specified by some regulations.

III. SCHEDULE OF TESTS

1. GENERAL CONSIDERATIONS

(a) OBJECT AND LIMITATIONS

The present investigation, in point of method of testing and types of protection tested, applies particularly to the interior columns of a structure. Due to greater exposure and smaller amount of protection, these form, in general, a more critical element in the column strength of a building as affected by fire than do the wall columns.

The size of the test columns in the present series is representative of those present in buildings of moderate height and of those under upper floors in higher buildings. The sizes involved were deemed to be best adapted for the initial investigation of the many variables pertaining to the fire resistance of building columns as designed and protected according to methods of modern building practice. Subsequent investigation of columns designed for the

heavier loads should be materially simplified by results of tests with columns of the proportions chosen for the present series.

The number of variables presented by the variety in material and form of columns, material and amount of protection, as well as the different methods of application, is greater than can be fully covered by a single series of tests, considering the limitations in time and expense incident with such an effort. It was necessary, therefore, to limit the investigation to the main forms of construction and protection, and in amount of protection, to the minimum and maximum as generally used.

(*b*) PRELIMINARY WORK ON SCHEDULE

This consisted of (1) investigation of existing methods of column design and column protection, including comparative study of the requirements of municipal building codes; (2) inquiry into field methods of erection and placing of columns and coverings; (3) study of the geographical distribution of production and use of materials for protective coverings of each type as an aid in selecting representative materials for the tests; (4) preparation of a preliminary schedule of tests which was submitted to engineers, contractors, architects, and public officers for criticisms and suggestions; (5) consideration of criticisms offered and formulation of a final testing program.

In amount of protection for interior columns, building ordinances require in general from 2 to 4 inch thicknesses of material outside of load-carrying elements, the requirements for the same grade of construction varying between these limits, as prescribed by different city codes. As it was considered desirable to determine the fire-resisting value of constructions in general use, it was decided to test protections of 2 and 4 inch commercial thicknesses, which, in connection with tests of unprotected and partly protected columns, as also of plaster on metal-lath protections, were deemed to include the general range of protection occurring in current building construction.

The schedule, as finally adopted at the conference of representatives of the cooperating units held in March, 1916, embodied many suggestions received in criticism of the preliminary schedule and was considered by all concerned to be the best procedure practicable, considering the extent of the field and the number of tests to be made.

2. SCHEDULE OF FIRE TESTS

(*a*) UNPROTECTED COLUMNS

Details of design of structural-steel, cast-iron, and pipe columns are shown in Figs. 1, 2, 3, 4, 6, and 7, and the calculated and applied working loads pertaining to each section are given in Table 41.

The unprotected structural-steel columns, which comprise one of each section type, are scheduled in Table 3*a*, together with the chief properties of their sections. These hold for the lower 12 feet, 8 inches of their length.

The list of unprotected cast-iron and pipe columns is given in Table 3*b*. Of the unprotected cast-iron columns, one was tested with ends restrained and three with unrestrained ends. Of the latter, one column, No. 11, was filled with concrete to increase its fire resistance. They were all horizontally cast, except No. 10A, which was a cast-iron pipe column.

The tests with pipe columns comprise one with a standard 7-inch pipe having plain concrete fill and one with an 8-inch pipe reinforced in the fill with structural angles.

(*b*) PARTLY PROTECTED COLUMNS

These include nine structural steel columns partly protected by filling the reentrant portions or interior with concrete and are scheduled in Table 3*c*, where also are shown the detail sections.

The proportions of the concrete mixtures used throughout this investigation are based on volume parts of the respective materials, Portland cement, sand, and coarse aggregate, taken in the given order. One bag of Portland cement weighing 94 pounds net was taken to be 1 cubic foot. The sand and coarse aggregate were measured by volume in the condition of density incident with shoveling them into the measures. The concrete sands as used had an average moisture content of 3 per cent. The Portland cement used in all tests of the whole series, except in those with pipe columns, was supplied from a mill in the Chicago district.

(*c*) COLUMNS PROTECTED BY PLASTER ON METAL LATH

Details of protection are given in Table 3*d*. The metal lath in Nos. 23, 25, and 26 was No. 24 expanded metal weighing 3.4 pounds per square yard including paint. That for test No. 24 was of 0.046-inch diameter wire, woven into ⅜-inch square mesh and weighing, painted, 3.2 pounds per square yard. The ribbed, expanded-metal lath in No. 27 had ribs ¾ inch high spaced 3½ inches apart, the weight being 7.9 pounds per square yard.

TABLE 3a.—SCHEDULE OF FIRE TESTS
Unprotected Structural Steel Columns
l= Effective Length, 12 ft. 8 in.

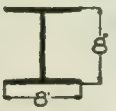
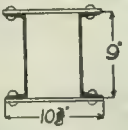
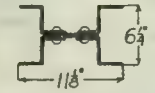
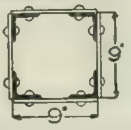
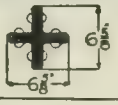
Test No.	SECTION	SECTION MEMBERS	Nominal Area, Sq. In.	Least Radius of Gyration, r, In.	$\frac{l}{r}$
1	Rolled H 	Solid Rolled H, 8 in.—34½ lb..	10.17	2.01	75.6
2	Plate and Angle	1 Plate, ½ by 6 in..... 4 Angles, 3 by 2½ by ½ in.....	13.00	1.36	111.8
3	Plate and Channel	2 Plates, ¼ by 8 in..... 2 Channels, 6 in.—8 lb.....	8.76	2.35	64.7
4	Latticed Channel 	2 Channels, 9 in.—13¼ lb..... Single lattice, ⅜ by 2 in.....	7.78	3.43	44.0
5	Z-bar and Plate 	1 Plate, ¼ by 5¾ in..... 4 Z-bars, 3 by ¼ in.....	9.32	1.86	81.7
6	I-beam and Channel	1 I-beam, 7 in.—15 lb..... 2 Channels, 7 in.—9¾ lb.....	10.12	2.11	72.1
7	Latticed Angle 	4 Angles, 3 by 3 by ⅜ in..... Single lattice, ¼ by 2¼ in.....	8.44	3.73	40.7
8	Starred Angle 	4 Angles, 3 by 3 by ⅜ in..... 1 Plate, ⅜ by 6⅝ in..... 2 Plates, ⅜ by 3⅛ in.....	13.27	1.40	108.8

TABLE 3b.—SCHEDULE OF FIRE TESTS

Unprotected Cast Iron and Pipe Columns

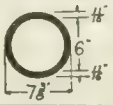
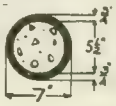
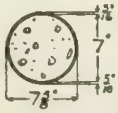
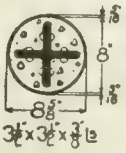
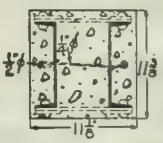
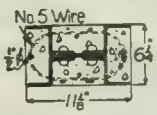
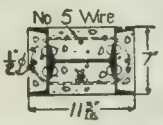
Test No.	SECTION	DETAILS	Nominal Area, Sq. In.	Effective Length, l, In.	Least Radius of Gyration, r, In.	$\frac{l}{r}$
9	Round Cast Iron, Horizontally cast	Ends restrained	14.73	152	2.23	68.2
10	Round Cast Iron, Horizontally cast	Ends not restrained	14.73	152	2.23	68.2
10-A	Round Cast Iron, Vertically cast 	Ends not restrained	14.45	150¼	2.38	63.2
11	Round Cast Iron, Horizontally cast, Concrete filled 	Ends not restrained Concrete, 1:2:4 Portland cement Joliet sand Joliet gravel	14.73	152	2.23	68.2
12	Steel Pipe, Concrete filled 	Ends not restrained Concrete, 1:1½:3 Portland cement Cambridge sand Westfield blue trap	Steel 6.93 Concrete 38.7	149	2.34	63.9
13	Reinforced Steel Pipe, Concrete filled 	Ends not restrained 4 angles inside pipe, 3½ in. 3½ in. ⅝ in. Concrete same as for No. 12	Steel 18.36 Concrete 40.1	152¾	2.24	68.2

TABLE 3c.—SCHEDULE OF FIRE TESTS

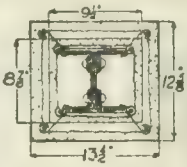
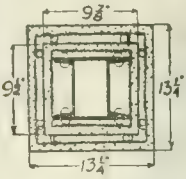
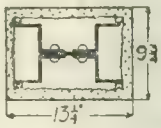
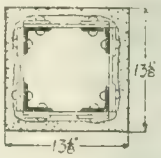
Columns Partly Protected by Concrete

Test No.	SECTION	PROTECTION	
		Mixture	Material
14	Rolled H	1:2:4	Portland cement Joliet sand Joliet gravel
15	Rolled H	1:2:4	Portland cement Plum Island sand Rockport granite
16	Plate and Angle	1:2:4	Portland cement Long Island sand New York trap
17	Plate and Angle	1:1½:4½	Portland cement Long Island sand Hard coal cinders
18	Latticed Channel 	1:2:4	Portland cement Long Island sand New York trap
19	Z-bar and Plate 	1:3:5	Portland cement Fox River sand Chicago limestone
20	I-beam and Channel 	1:3:5	Portland cement Long Island sand New York trap
21	I-beam and Channel	1:3:5	Portland cement Long Island sand New York trap
22	Latticed Angle	1:2:4	Portland cement Fox River sand Chicago limestone

NOTE: The horizontal ties, consisting of ¼-inch bars or No. 5 wire, are bent around or wired to the vertical ½-inch bars, and are spaced 18 inches apart vertically.

TABLE 3d.—SCHEDULE OF FIRE TESTS

Columns Protected by Plaster on Metal Lath

Test No.	SECTION	*PROTECTION
23	Plate and Angle 	Two 2-coat layers of Portland cement plaster on No. 24 expanded metal lath, each layer 1 in. thick, with a ¾-in. air space between layers
24	Plate and Channel 	Two 2-coat layers of Portland cement plaster on woven wire lath, ¾-in. mesh, each layer ⅞ in. thick, with a ¾-in. air space between layers
25	Z-bar and Plate 	One 2-coat layer of Portland cement plaster, 1 in. thick, on No. 24 expanded metal lath
26	Latticed Angle 	One 2-coat layer of Portland cement plaster, 1⅛ in. thick, on No. 24 expanded metal lath
27	Round Cast Iron Column Section same as No. 9	One 2-coat layer of Portland cement plaster, 1½ in. thick, on high ribbed expanded metal lath with ½-in. broken air space

*The plaster consisted of 1 part Portland cement, 1-10 part hydrated lime, 2½ parts coarse lake sand. Hair was used in the first coat; the second coat was trowelled smooth.

The proportion of the plaster was 1 part Portland cement, $\frac{1}{10}$ part hydrated lime, and 2½ parts lake sand of medium grade of coarseness, all materials being measured by loose volume. The sand had an average moisture content of 3 per cent.

The thickness of the layers was measured from the inside of the lath. For Nos. 23, 24, and 25 a layer thickness of 1 inch was desired and was attained in Nos. 23 and 25. In No. 24 the layer thickness averaged ⅞ of an inch. In No. 26 an attempt was made to place a 2-inch thick layer, but this proved impracticable with the given section using two body coats, an average thickness of 1⅛ inches being finally attained. In No. 27 an effort was made to make a solid covering by pushing the plaster through the lath against the outside of the column. This succeeded only partly, a broken air space averaging ½ inch in depth remaining between the covering and the column.

For all plaster on metal-lath protections the extreme edges of the bracket angles near the top of the column were covered by a single layer 1 inch thick.

The coverings were finished by troweling and floating the second coat to a smooth surface.

(*d*) COLUMNS PROTECTED BY CONCRETE

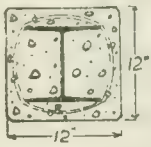
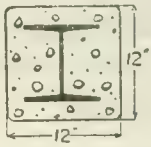
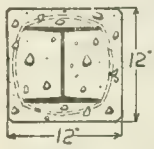
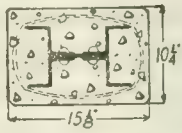
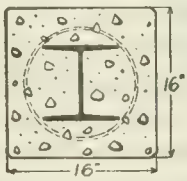
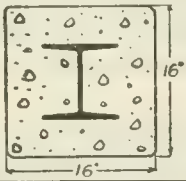
The tests with concrete protections are described in Table 3*e*.

Six combinations of fine and coarse concrete aggregates as used in building construction in four large industrial centers are included. The aggregate combinations and districts represented are (1) Rockport granite with Plum Island sand for the Boston (Mass.) district; (2) Chicago limestone with Fox River sand and Joliet gravel with Joliet sand for the Chicago (Ill.) district; (3) Cleveland sandstone with Pelee Island sand for the Cleveland (Ohio) district; (4) New York trap rock with Long Island sand and hard-coal cinders with Long Island sand for the New York (N. Y.) district. The districts were chosen so as to obtain representation for the main groups of rocks that are used as concrete aggregate. The aggregates are further described in paragraph 6 of Section V (p. 61).

The proportions of the mixtures used were 1 : 2 : 4 and 1 : 3 : 5 for the stone and gravel concrete, and for the cinder concrete, 1 : 1½ : 4½ and 1 : 2 : 5. The cinders were used unscreened as received, except that pieces larger than 1 inch were crushed to smaller size.

TABLE 3e.—SCHEDULE OF FIRE TESTS

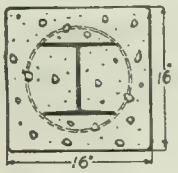
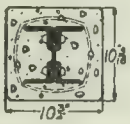
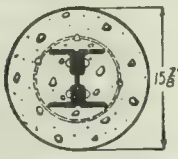
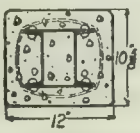
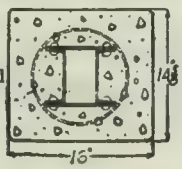
Columns Protected by Concrete

Test No.	SECTION	PROTECTION		
		Thickness, In.	Mixture	Material
28	Rolled H 	2	1:2:4	Portland cement Fox River sand Chicago limestone
28 A	Rolled H 	2	1:2:4	Portland cement Fox River sand Chicago limestone
29	Rolled H	2	1:2:4	Portland cement Long Island sand New York trap
30	Rolled H 	2	1:2:4	Portland cement Joliet sand Joliet gravel
31	Rolled H	2	1:2:4	Portland cement Pelee Island sand Cleveland sandstone
32	Rolled H	2	1:2:5	Portland cement Long Island sand Hard coal cinders
32 A	Z-bar and Plate 	2	1:2:5	Portland cement Long Island sand Hard coal cinders
33	Rolled H 	4	1:2:4	Portland cement Fox River sand Chicago limestone
33 A	Rolled H 	4	1:2:4	Portland cement Fox River sand Chicago limestone

NOTE: Ties where used are of No. 5, B. & S. gage, steel wire, wound spirally on a pitch of 8 in.

TABLE 3e.—SCHEDULE OF FIRE TESTS

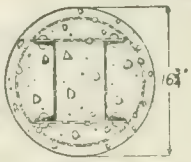
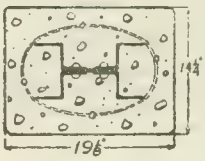
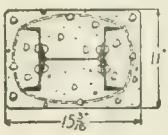
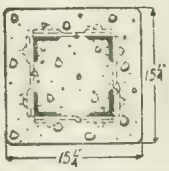
Columns Protected by Concrete—Continued

Test No.	SECTION	PROTECTION		
		Thickness, In.	Mixture	Material
34	Rolled H 	4	1:2:4	Portland cement Plum Island sand Rockport granite
34A	Rolled H	4	1:2:4	Portland cement Plum Island sand Rockport granite
35	Rolled H	4	1:3:5	Portland cement Fox River sand Chicago limestone
36	Plate and Angle 	2	1:2:4	Portland cement Long Island sand New York trap
37	Plate and Angle 	4	1:2:4	Portland cement Long Island sand New York trap
38	Plate and Channel 	2	1:2:4	Portland cement Joliet sand Joliet gravel
39	Plate and Channel 	4	1:2:4	Portland cement Meramec River sand Meramec River gravel

NOTE: Wire ties are of No. 5, B. & S. gage, steel wire, wound spirally on a pitch of 8 in.

TABLE 3e.—SCHEDULE OF FIRE TESTS

Columns Protected by Concrete—Concluded

Test No.	SECTION	PROTECTION		
		Thickness In.	Mixture	Material
40	Latticed Channel 	2	1:2:4	Portland cement Long Island sand New York trap
41	Z-bar and Plate 	4	1:3:5	Portland cement Fox River sand. Chicago limestone
42	Z-bar and Plate	4	1:3:5	Portland cement Fox River sand Chicago limestone
43	I-beam and Channel 	2	1:2:4	Portland cement Pelee Island sand Cleveland sandstone
44	I-beam and Channel	2	1:3:5	Portland cement Pelee Island sand Cleveland sandstone
45	Starred Angle	2	1:2:4	Portland cement Meramec River sand Meramec River gravel
46	Latticed Angle 	2-in. outside rivets. 3½-in. outside angles.	1:2:4	Portland cement Long Island sand New York trap
47	Round Cast Iron	2	1:2:5	Portland cement Long Island sand Hard coal cinders

NOTE: Wire ties where used are of No. 5, B. & S. gage, steel wire wound spirally on a pitch of 8 in.

Ties consisting of No. 5 (B. and S. gage) bright basic steel wire were wound spirally around the structural section on a vertical pitch of 8 inches. The tie was omitted in Test No. 28A and 33A in order to determine what effect, if any, it has on the effectiveness of the covering. In No. 46 it was omitted because the latticed section was deemed to afford sufficient support for the outside covering. In No. 47 an attempt was made to place the covering with the tie wire supported on 1-inch T bars. This proved impracticable on account of the obstruction it made to the flow of the concrete, this covering being, therefore, placed without tie.

For the square coverings the thickness was measured, respectively, from the face of the flange and from its extreme edge. For the round protections, test Nos. 37 and 40, the thickness of covering on the face of the flange was greater than the given nominal thickness and that on the flange edge smaller than the given thickness, each by about 5/8 of an inch.

For the concrete, as well as all other full protections, the thickness of covering on the extreme edges of the bracket angles near the top of the column was 1 inch.

(*e*) COLUMNS PROTECTED BY HOLLOW CLAY TILE

The tests with hollow clay-tile protections are given in Table 3*f*.

Five kinds of tile from as many producing regions were used for the coverings. These include two varieties of surface-clay tile, one of shale, and two of semifire clay. They are further described in paragraph 11 of Section V (p. 73).

The tile was set in mortar consisting of 1 part, by loose volume, of Portland cement, 1 part of lime putty, and 4 parts of fine beach or bank sand. The sand had an average moisture content of 4 per cent. The thickness of mortar between the tile and the flanges of the test columns varied between the different protections from 1/2 to 1 1/8 inches. The thickness of horizontal joints between the tile courses averaged 3/8 of an inch where no wire mesh was used and 5/8 of an inch where used. The vertical mortar joints varied in thickness from 1/4 to 3/4 of an inch, depending on the design of the covering.

The upper tile courses were set out sufficiently to allow the extreme edges of the bracket angles to be covered by a 1-inch thickness of tile and mortar.

Two forms of mechanical ties for the tile were used. One consisted of a No. 12 (B. and S. gage) iron wire, tied around the outside of each course, and the other of strips of 3/8-inch wire mesh (diameter of wire, 0.046 inch) laid in the horizontal joints and lapped at the corners.

TABLE 31.—SCHEDULE OF FIRE TESTS

Columns Protected by Hollow Clay Tile

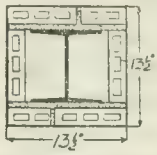
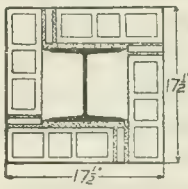
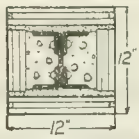
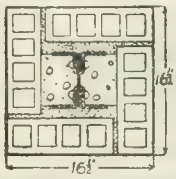
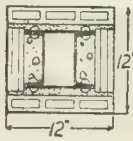
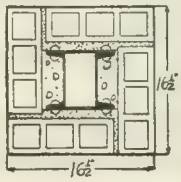
Test No.	SECTION	Thickness of Tile, In.	Kind of Tile and Method of Application	Filling
48	Rolled H 	2	Semi-fire clay, New Jersey district ¾-in. mortar on flanges Outside wire ties	No filling
49	Rolled H 	4	Semi fire clay, New Jersey district ¾-in. mortar on flanges	No filling
50	Plate and Angle 	2	Surface clay, Boston district ¾-in. mortar on flanges	1:3:5 concrete Portland cement Plum Island sand Rockport granite
50-A	Plate and Angle	2	Same as No. 50	Same as No. 50
51	Plate and Angle 	4	Surface clay, Boston district 1⅛-in. mortar on flanges Outside wire ties	1:3:5: concrete Portland cement Plum Island sand Rockport granite
51-A	Plate and Angle	4	Same as No. 51	Same as No. 51
52	Plate and Channel 	2	Ohio shale ¾-in. mortar on flanges Outside wire ties	1:2:5 concrete Portland cement Long Island sand Hard coal cinders
53	Plate and Channel 	4	Ohio shale 1-in. mortar on flanges Outside wire ties	1:2:5: concrete Portland cement Long Island sand Hard coal cinders

TABLE 31.—SCHEDULE OF FIRE TESTS

Columns Protected by Hollow Clay Tile—Continued

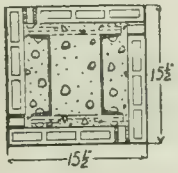
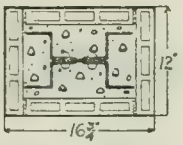
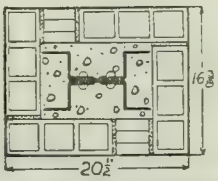
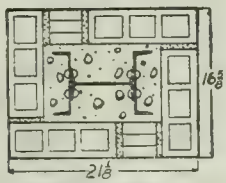
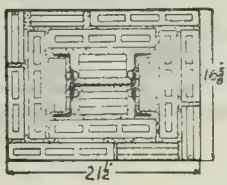
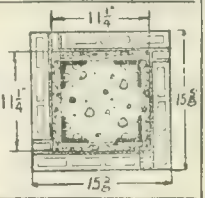
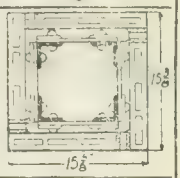
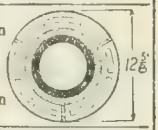
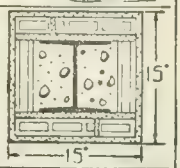
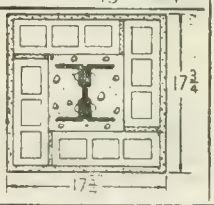
Test No.	SECTION	Thickness of Tile, In.	Kind of Tile and Method of Application	Filling
54	Latticed Channel 	2	Ohio semi-fire clay Outside wire ties	1:3:5 concrete. Portland cement Long Island sand New York trap
55	Z-bar and Plate 	2	Ohio semi-fire clay Outside wire ties	1:3:5: concrete, Portland cement Fox River sand Chicago limestone
56	Z-bar and Plate 	4	Ohio semi-fire clay ⅜-in. wire mesh in horizontal joints	1:3:5: concrete, Portland cement Fox River sand Chicago limestone
57	I-beam and Channel 	4	Surface clay, Chicago district Outside wire ties	1:3:5: concrete, Portland cement Fox River sand Chicago limestone
58	I-beam and Channel 	Two 2-in.	Surface clay, Chicago district ½-in. mortar on flanges ⅜-in. wire mesh in horizontal joints	Hollow clay tile. 2 by 12 by 6 in. at at ends 3 by 12 by 6 in. at sides
59	I-beam and Channel	Two 2-in.	Surface clay, Chicago district ½-in. mortar on flanges Outside wire ties	Hollow clay tile, 2 by 12 by 6 in. at ends 3 by 12 by 6 in. at sides

TABLE 31.—SCHEDULE OF FIRE TESTS

Columns Protected by Hollow Clay Tile—Concluded

Test No.	SECTION	Thickness of Tile, In.	Kind of Tile and Method of Application	Filling
60	Latticed Angle 	2	Ohio semi-fire clay ¼-in. mortar between fill and tile Outside wire ties	1:2:4: concrete Portland cement Long Island sand New York trap Fill placed before tile was set
61	Latticed Angle 	2	Ohio semi-fire clay Outside wire ties	No filling
62	Round Cast Iron 	2	Porous semi-fire clay, New Jersey district ¾-in. mortar Outside wire ties	No filling
63	Round Cast Iron	2	Same as No. 62	No filling
76	Rolled H 	2	Ohio shale: Ohio semi-fire clay; semi-fire clay, New Jersey district ¾-in. mortar on flanges ⅜-in. wire mesh in horizontal joints Tile covered with a 2-coat, ¾-in. layer of 1:3 gypsum plaster	1:3:5: concrete Portland cement Fox River sand Chicago limestone
77	Plate and Angle 	4	Semi-fire clay, New Jersey; surface clay, Chicago; surface clay, Boston district 1⅛-in. mortar on flanges ⅜-in. wire mesh in horizontal joints Tile covered with a 2-coat, ⅝-in. layer of 1:2½ lime plaster	1:3:5 concrete Portland cement Fox River sand Chicago limestone

NOTE: The mortar used in setting the tile consisted of 1 part Portland cement, 1 part lime putty and 4 parts fine beach or bank sand.

The filling inside of the tile, where used, consisted of concrete or hollow clay tile. The concrete was placed after the tile was set, except in case of No. 60, where the fill was placed and allowed to harden before setting the tile.

All protections were tested in the unplastered condition except those of Nos. 76 and 77. The former was plastered with a two-coat layer of gypsum plaster ¾ inch thick. The first coat of about ⅝ inch thickness consisted of 1 volume part neat, fibered, calcined gypsum and 3 parts fine lake sand, and the finish coat, 1 volume part neat, unfibered, calcined gypsum and 2 parts hydrated lime. No. 77 was plastered with lime plaster ⅝ of an inch in thickness, the first coat consisting of 1 volume part slaked-lime putty and 2½ parts of fine lake sand, the finish coat being the same as for No. 76.

(*f*) COLUMNS PROTECTED BY GYPSUM BLOCK

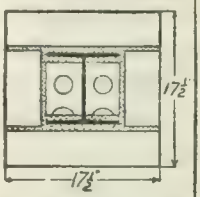
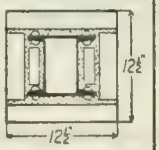
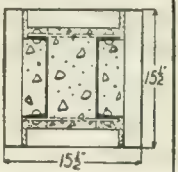
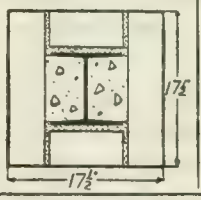
The tests with gypsum block protections scheduled in Table 3*g* consist of two with 2-inch and three with 4-inch solid block. The material was supplied from two factories, one located in the middle-western section of the country and the other in the eastern section. The proportion of the mortar used for setting the block was 1 part, by volume, of neat, unfibered, calcined gypsum, and 3 parts, by volume, of fine lake sand. The latter as used had a moisture content of about 3 per cent.

In Nos. 64 and 65 the blocks were tied with No. 22 corrugated, galvanized-iron strips, ¾ of an inch wide by 6 inches long, placed in the horizontal joints and across all vertical joints, one over each joint in the 2-inch covering and two over each joint in the 4-inch covering. In Nos. 66, 67, and 67A strips of woven wire (0.046-inch wire diameter) of ⅜-inch mesh were laid in the horizontal joints over all vertical joints. The size of the strips for the 2-inch protection was 2 by 14½ inches and for the 4-inch protections, 3½ by 16½ inches, the strips being laid so the outer edges were ½ inch from the surface of the covering. The thickness of horizontal joints averaged ½ inch between the 2-inch blocks, and ¾ inch between the 4-inch blocks. The vertical joints were about ⅜ of an inch thick.

The space between the blocks and column flanges was filled with mortar as the blocks were set. The remaining space inside of the blocks was filled with gypsum block set in mortar in the

TABLE 3g.—SCHEDULE OF FIRE TESTS

Columns Protected by Gypsum Block

Test No.	SECTION	Thickness of Block, In.	Kind of Gypsum and Method of Application	Filling
64	Rolled H 	4	Western gypsum (solid) ¾-in. mortar on flanges Corrugated wall ties in horizontal joints	Hollow Western gypsum block
65	Plate and Channel 	2	Western gypsum (solid) 1-in. mortar on flanges Corrugated wall ties in horizontal joints	Solid Western gypsum block
66	Latticed Channel 	2	Eastern gypsum (solid) ¾-in. mortar on lattice ⅜-in. wire mesh in horizontal joints	1:1:4, Calcined gypsum Fine lake sand Broken gypsum block
67	Rolled H 	4	Eastern gypsum (solid) ¾-in. mortar on flanges ⅜-in. wire mesh in horizontal joints	1.1:4, Calcined gypsum Fine lake sand Broken gypsum block
67-A	Rolled H	4	Same as No. 67	Same as No.67

NOTE: The gypsum blocks were set in mortar consisting of 1 part by volume of neat unfibered calcined gypsum and 3 parts fine lake sand.

case of Nos. 64 and 65. For the other gypsum protections the filling material consisted of 1 part, by volume, of neat, unfibered, calcined gypsum, 1 part fine lake sand, and 4 parts gypsum block broken to maximum size of 2 inches, the whole being mixed to wet consistency.

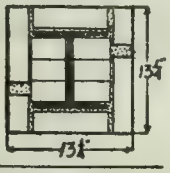
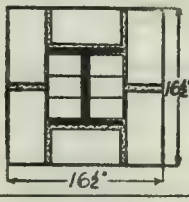
The methods of tying the blocks and filling within them conform with the recommendations of the manufacturers by whom they were supplied.

(*g*) COLUMNS PROTECTED BY BRICK

The two tests with brick protection are described in Table 3*h*. The proportion of the mortar was the same as that used for the clay-tile protections. In placing the brick no metal ties were used, the brick being in each case set so as to obtain the best bond possible with the given design of covering. The thickness of the horizontal joints averaged ½ inch and that of the vertical joints varied from ¼ to 1¼ inches.

TABLE 3h.—SCHEDULE OF FIRE TESTS

Columns Protected by Brick

Test No.	SECTION	Thickness of Brick, In.	Kind of Brick	Filling
68	Rolled H 	2¼	Chicago common brick set on edge and end ⅝-in. mortar on flanges	Chicago common brick
69	Rolled H 	3¾	Chicago common brick laid flat ⅝-in. mortar on flanges	Chicago common brick

NOTE: The mortar consisted of 1 part Portland cement, 1 part lime putty and 4 parts bank sand.

(*h*) REINFORCED-CONCRETE COLUMNS

These are scheduled in Table 3*i* and details of design are given in Figs. 8 and 9 (p. 26-27).

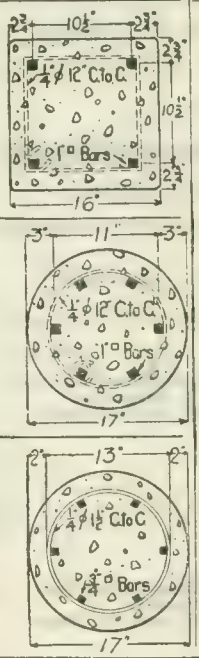
A thickness of 2 inches of the concrete next to the surface of the column is taken as a protective covering for the concrete and steel within it and is not included in the given effective areas.

TABLE 3i.—SCHEDULE OF FIRE TESTS

Reinforced Concrete Columns

Effective Length, 12 ft. 8 in.

Test No.	SECTION	Mix	Material	*Effective Area, Sq. In.	Reinforcement, Percent of Effective Area
70	Square Vertically Reinforced	1:2:4	Portland cement Fox River sand Chicago limestone	Concrete, 140 Steel, 4.00	Vertical, 2.78 Lateral, 0.14
71	Square Vertically Reinforced	1:2:4	Portland cement Long Island sand New York trap	Same as No. 70	Same as No. 70
72	Round Vertically Reinforced	1:2:4	Portland cement Fox River sand Chicago limestone	Concrete, 127 Steel, 6.00	Vertical, 4.52 Lateral, 0.13
73	Round Vertically Reinforced	1:2:4	Portland cement Long Island sand New York trap	Same as No. 72	Same as No. 72
74	Hooped Reinforced	1:2:4	Portland cement Fox River sand Chicago limestone	Concrete, 129 Steel, 3.38	Vertical, 2.54 Lateral, 0.99
75	Hooped Reinforced	1:2:4	Portland cement Long Island sand New York trap	Same as No. 74	Same as No. 74



*A depth of 2 in. all around deducted from the gross section for fire protection.

(*i*) TIMBER COLUMNS

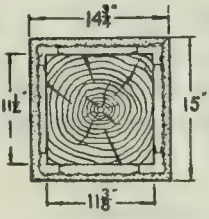
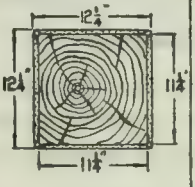
The schedule of timber columns is given in Table 3*j* and details of columns and bearings are shown in Fig. 10. The tests include two species of timber, each tested with two types of bearing details.

No. 78 was protected by a single layer of plaster on metal lath, the details of application being the same as for the protections listed in Table 3*d*. No. 80 was covered with gypsum wall board 3/8 inch thick, nailed to the timber at the corners with No. 4 lathing nails on 2-inch centers, and finished with kalsomine. The other timber columns were tested unprotected.

TABLE 3j.—SCHEDULE OF FIRE TESTS

Timber Columns

Effective Length, 12 ft. 8½ in.

Test No.	SPECIES	SECTION	Protection of Column and Cap	Bearing Details
78	Longleaf pine		One 2-coat layer of Portland cement plaster, 1 in. thick, on woven wire lath, ⅜-in. mesh, ¼-in. air space. Plaster consisted of 1 part Portland cement, 1/10 part hydrated lime, 2½ parts coarse lake sand	Cast iron cap and pintle
79	Longleaf pine		Unprotected	Cast iron cap and pintle
80	Longleaf pine		One thickness of ⅜-in. gypsum wall board with metal corner beads, nailed to column	Steel plate cap and 8 in. long timber strut.
81	Longleaf pine		Unprotected	Steel plate cap and 6 in long timber strut
82	Douglas fir		Unprotected	Cast iron cap and pintle
83	Douglas fir		Unprotected	Steel plate cap and 18 in. long timber strut

3. SCHEDULE OF FIRE AND WATER TESTS

This series was introduced in order to determine the effect on coverings and columns of the impact and sudden cooling produced by hose streams applied to them when in a highly heated condition.

In order to introduce all of the materials used in the fire-test series with a minimum number of tests, two or three kinds of material of a given class were applied in each test.

(*a*) COLUMNS PROTECTED BY CONCRETE

The fire and water tests with concrete protections are scheduled in Table 4*a*. Three kinds of concrete were applied to each column. In this and succeeding tables the concrete is distinguished by the name of the coarse aggregate, the sand used with each being the same as in the combinations given above in paragraph 2*d* (p.37).

The metal tie in the covering is the same as for the corresponding protections in the fire-test series. In No. 102 the tie is omitted.

(*b*) COLUMNS PROTECTED BY HOLLOW CLAY TILE

These coverings are also placed in three sections (Table 4*b*) with one of the varieties of tile used in the protections of the fire-test series in each section. The proportions of the mortar and size of metal ties are the same as for the corresponding protections of the fire-test series.

(*c*) COLUMNS PROTECTED BY GYPSUM BLOCK

The two kinds of gypsum with the method of application peculiar to each, described in paragraph 2*f* of this section are employed in the two tests (Table 4*c*). The filling for both consists of a 1 : 1 : 4 mixture by volume parts of calcined gypsum, fine lake sand, and broken gypsum block, mixed to wet consistency.

(*d*) PLASTER ON METAL-LATH PROTECTION

The protection of this type included in the fire and water series is described in Table 4*d*. The metal lath, proportion of the plaster, and method of application are the same as for the corresponding protections of the fire-test series.

(*e*) REINFORCED-CONCRETE COLUMNS

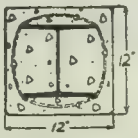
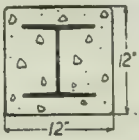
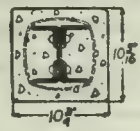
One column of each type is included in this series. Details of concrete and reinforcement are given in Table 4*e*.

(*f*) UNPROTECTED CAST-IRON COLUMNS

Two duplicate columns are listed in Table 4*f*. They are vertically cast and have bearing details as shown in Fig. 6 (*c*) (p. 22).

TABLE 4a.—SCHEDULE OF FIRE AND WATER TESTS

Columns Protected by Concrete

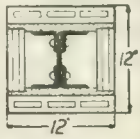
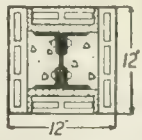
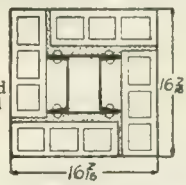
Test No.	SECTION	PROTECTION		
		Thickness, In.	Mixture	*Kind of Concrete
101	Rolled H 	2	1:2:4	Chicago limestone New York trap Joliet gravel
102	Rolled H 	2	1:2:4	New York trap Joliet gravel Chicago limestone
103	Plate and Angle	4	1:2:4	New York trap New England granite Chicago limestone
104	Plate and Angle 	2	1:2:5 1:2:4 1:2:4	Hard coal cinders Cleveland sandstone New York trap

*Three kinds of concrete were used on each column, placed in three vertical sections in the order named, from top to bottom of column.

NOTE: Ties where used are of No. 5, B. & S. gage, steel wire, wound spirally on a pitch of 8 in.

TABLE 4b.—SCHEDULE OF FIRE AND WATER TESTS

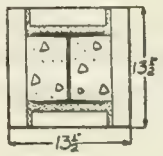
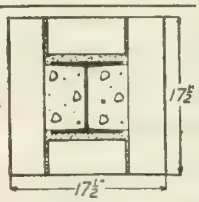
Columns Protected by Hollow Clay Tile

Test No.	SECTION	Thickness of Tile, In.	*Kind of Tile and Method of Application	Filling
105	Plate and Angle 	2	Surface clay, Boston district; semi-fire clay, N. J. district; Ohio shale ⅞-in. mortar on flanges Outside wire ties	No filling
106	Plate and Angle 	2	Ohio semi-fire clay; surface clay, Chicago district; Ohio semi-fire clay ⅞-in. mortar on flanges Outside wire ties on upper half ⅜-in. wire mesh in horizontal joints in lower half	1:3:5 concrete, Portland cement Long Island sand Chicago limestone
107	Plate and Channel 	4	Ohio shale; semi-fire clay, N. J. district; surface clay, Boston district 1-in. mortar on flanges Outside wire ties on upper half ⅜-in. wire mesh in horizontal joints in lower half	No filling

*Three kinds of tile were used on each column, placed in three vertical sections in the order named from top to bottom of column.

TABLE 4c.—SCHEDULE OF FIRE AND WATER TESTS

Columns Protected by Gypsum Block

Test No.	SECTION	Thickness of Block, In.	*Kind of Gypsum and Method of Application	Filling
108	Rolled H 	2	Western gypsum (solid); Eastern gypsum (solid) ¾-in. mortar on flanges Corrugated wall ties in horizontal joints in upper half ⅜-in. wire mesh in horizontal joints in lower half	1:1:4 Calcined gypsum Fine lake sand Broken gypsum block
109	Rolled H 	4	Eastern gypsum (solid); Western gypsum (solid) ¾-in. mortar on flanges ⅜-in. wire mesh in horizontal joints in upper half. Corrugated wall ties in horizontal joints in lower half	1:1:4 Calcined gypsum Fine lake sand Broken gypsum block

*Two kinds of block were used on each column, placed in two vertical sections in the order named from top to bottom.

TABLE 4d.—SCHEDULE OF FIRE AND WATER TESTS

Column Protected by Plaster on Metal Lath

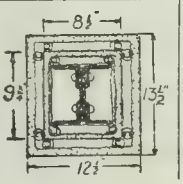
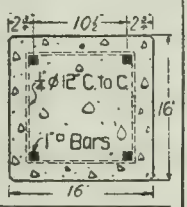
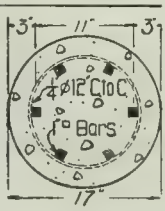
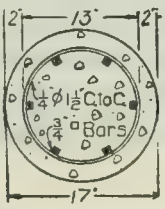
Test No.	SECTION	PROTECTION
110	Plate and Angle 	Two 2-coat layers of Portland cement plaster; inner layer ⅜ in. thick, on woven wire lath; outer layer 1 in. thick, on expanded metal lath. ¾-in. air space between layers. Proportion of plaster, 1 part Portland cement, 1/10 part hydrated lime, 2½ parts coarse lake sand

TABLE 4e.—SCHEDULE OF FIRE AND WATER TESTS

Reinforced Concrete Columns

Effective Length, 12 ft. 8 in.

Test No.	SECTION	Mixture	*Kind of Concrete	**Effective Area, Sq. In.	Reinforcement, Percent of Effective Area
111	Square Vertically Reinforced 	1:2:4	Chicago limestone Meramec R. gravel Chicago limestone	Concrete, 140 Steel, 4.00	Vertical, 2.78 Lateral, 0.14
112	Round Vertically Reinforced 	1:2:4	Chicago limestone Meramec R. gravel Joliet gravel	Concrete, 127 Steel, 6.00	Vertical, 4.52 Lateral, 0.13
113	Hooped Reinforced 	1:2:4	New York trap Meramec R. gravel Rockport granite	Concrete, 129 Steel, 3.38	Vertical, 2.54 Lateral, 0.99

*Three kinds of concrete were used in each column, placed in three sections in the order named, from top to bottom of column.

**A depth of 2 in. all around deducted from the gross section for fire protection.

TABLE 4f.—SCHEDULE OF FIRE AND WATER TESTS

Unprotected Cast Iron Columns

Effective Length, 12 ft. 6¼ in.

Test No.	SECTION	DETAILS	Nominal Area, Sq. In.	Least Radius of Gyration, r In.	$\frac{l}{r}$
114	Round Cast Iron, Vertically cast	Ends not restrained	14.45	2.38	63.2
115	Round Cast Iron, Vertically cast	Ends not restrained	14.45	2.38	63.2

IV. PLACING OF COVERINGS AND CONCRETE COLUMNS

The work was planned so as to reproduce as nearly as possible the conditions obtaining in building construction in point of methods of application and workmanship. This was done to make the results of the tests applicable without undue allowance for differences that otherwise might be deemed to exist between the test sample and a similarly constructed member in a building.

1. CONCRETE PROTECTIONS AND COLUMNS

(*a*) FORMS AND STAGING

The wood forms were made of 1⅝-inch yellow-pine planks, with clamps spaced about 2 feet apart vertically. The round columns and coverings were cast in metal forms made of No. 12 (0.1094 inch thick) sheet steel, the forms being made into halves which were bolted together through angles riveted on their edges.

The form was supported within a staging extending to the top of the test column proper, the floor of the staging being used as a platform from which the concrete was placed and on which subsequently the form for the column head or column-head protection was erected. A view of the staging with column forms in place is shown in Fig. 11.

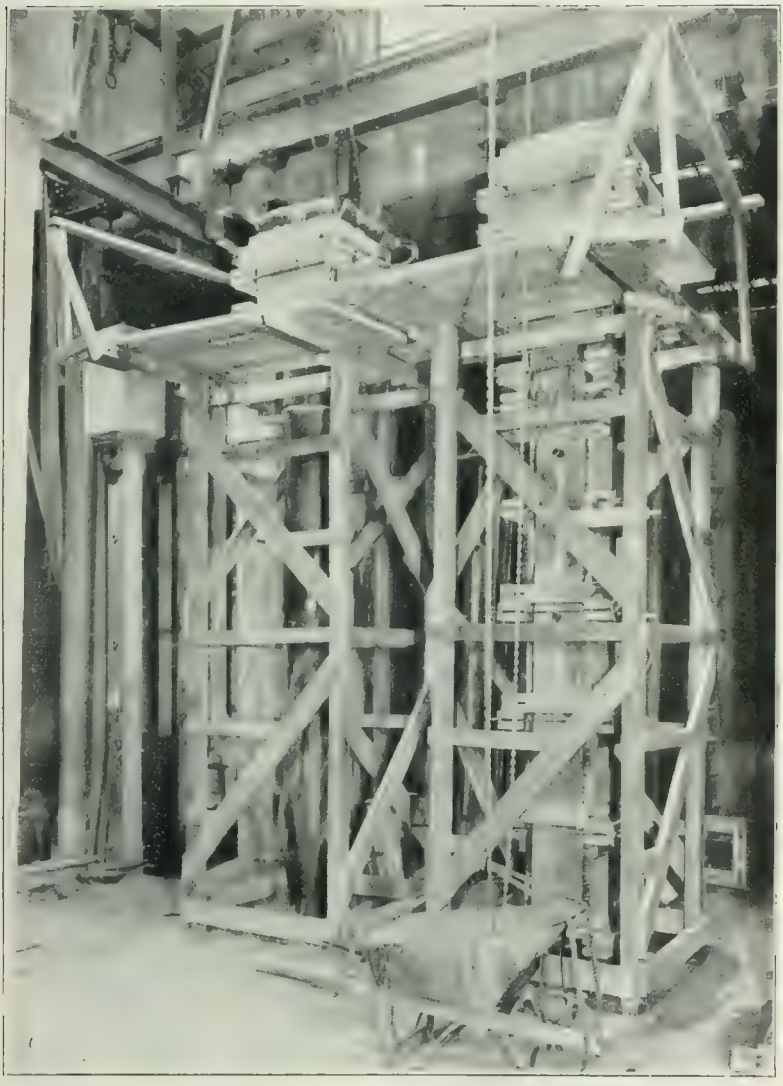


FIG. 11.—*Forms and staging for placing concrete*

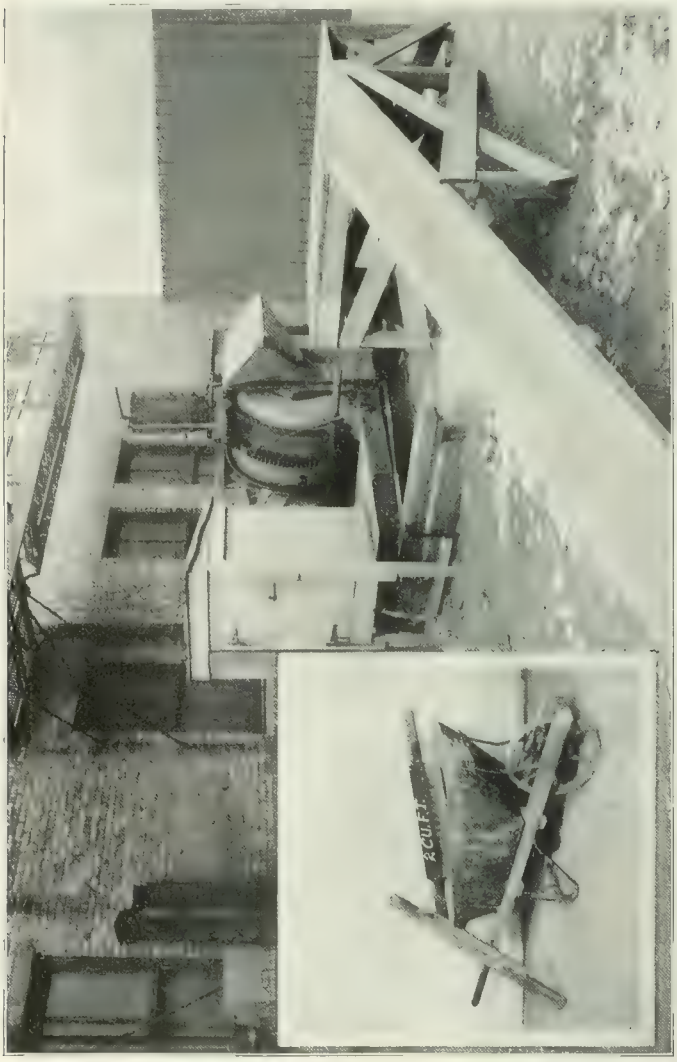


FIG. 12 *Concrete mixer*

(*b*) METHOD OF PROPORTIONING

The proportions were based on volume parts of the materials as taken from the bins, except that the Portland cement was measured in the original package, one bag, containing 94 pounds of cement, being taken to be 1 cubic foot. The sand and stone were measured in deep, steel wheelbarrows in 2 and 3 cubic-foot portions, the volume for each being determined by striking off the top with a board cut to the required shape (Fig. 12). In some tests, where the mixtures appeared lean in sand, two or three shovelfuls of sand were substituted for an equal amount of stone.

(*c*) MIXING AND PLACING

The concrete was mixed in a motor-driven batch mixer having a capacity of 6 cubic feet of mixed concrete. The materials were charged into the mixer in the following general order, subject to minor variations introduced by the different men in charge of the mixing: (1) 2 cubic feet coarse aggregate, (2) 2 or 3 cubic feet of sand, (3) 1 bag Portland cement, (4) 2 or 3 cubic feet of coarse aggregate, (5) water. Before admitting the water the materials were mixed dry for a period varying from ¼ to ½ minute, the total time of mixing being limited to 1½ minutes as a maximum and 1 minute as minimum. The water was measured by means of a gage glass and scale, the former connecting with a vertical measuring tank placed above the mixer (Fig. 12).

The concrete was discharged into wheelbarrows which were raised to the platform of the staging for discharge into the forms. The concrete was spaded along the inside of the form and the latter was tapped with a hand hammer to assist in obtaining a good concrete surface.

To obtain workmanship comparable with that on field-placed concrete, several experienced men connected with local construction companies were at different times placed in charge of the mixing and placing, about one-half of the total number of concrete coverings and columns being thus placed. The methods thus introduced were followed in the mixing and placing of the concrete for the remaining columns.

2. PLASTER ON METAL-LATH PROTECTIONS

(*a*) PLACING OF LATH

For test No. 23 the lath for both the outer and the inner layer was supported on round bars held upright by iron clips made from 1/8 by 1 inch flat bars and secured to the structural-steel section. This method proved very cumbersome, and the lath for the other steel columns was supported on 3/8 by 3/4 inch channels held in vertical position by wire ties. The high-ribbed lath was supported directly on the metal.

The lath was placed around the column in horizontal courses with laps of about 3 inches. Horizontal joints between sheets had generally shorter laps. All laps were wired with No. 18 wire ties placed 3 to 6 inches apart vertically and one on the middle of each side of the horizontal joints.

(*b*) APPLYING THE PLASTER

The plaster coats were of the maximum thickness practicable with the given mixture of plaster. The first coat of a layer was allowed to set two or three days before applying the second coat.

The lathing and plastering were done by experienced men obtained through a local plastering contractor.

3. HOLLOW CLAY-TILE AND BRICK PROTECTIONS

(*a*) PROPORTIONING OF MORTAR

The mortar was proportioned by volume parts of loose materials. The proportion used was 1 part Portland cement, 1 part stiff lime putty (slaked lump lime), and 4 parts fine bank or lake sand.

(*b*) PLACING OF TILE AND BRICK

The coverings were detailed in advance, and tile of the required size was supplied when possible. All cutting of tile and brick was done on the job with the hammer or trowel. A view of tile and gypsum-block protections under construction is shown in Fig. 13.

The work was done on a contract basis by a mason contractor, and it is thought that the workmanship obtained approximates that secured in good building practice.

(*a*) PLACING OF CONCRETE FILLING

Where concrete filling between the tile and column was used, the tile was held in place by clamping both ways every 2 feet, boards being placed along the protection inside of the clamps. The filling was placed from the platform of the staging shown in

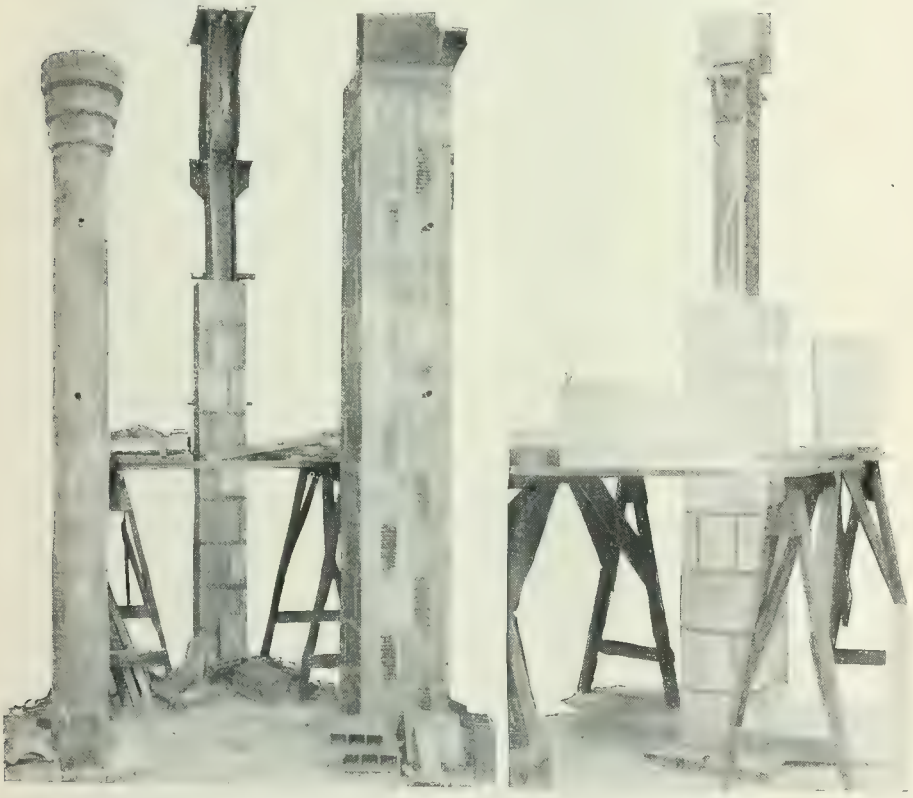


FIG. 13.—*Placing of clay-tile and gypsum-block protections*

Fig. 11, the whole space being filled in one continuous operation. The clamps effectively held the tile against the pressure of the wet fill, and very few mortar joints were broken from this cause.

4. GYPSUM-BLOCK PROTECTIONS

(*a*) PROPORTIONING OF MORTAR

The proportion of mortar used, 1 : 3, neat, unfibered, calcined gypsum and fine lake sand, was the richest mixture that would work satisfactorily under the trowel, the materials being measured by loose volume.

(*b*) PLACING OF BLOCK

The blocks were cut from standard-size partition blocks with a hand saw, the resulting pieces being generally all used either in the covering or filling. The mortar joints in the covering and between the blocks and flanges of the test column were well filled.

(*c*) PLACING OF FILLING

Where wet filling was used the dry materials were first turned three times by hand and then mixed with water in small batches. The filling was placed as the blocks were laid up, two courses being generally filled at one time.

The placing and filling of the gypsum coverings were done by a fireproofing contractor employing men experienced in handling the given material.

5. METHOD OF STORAGE

Normal air storage was used for all columns and for the greater number of auxiliary test specimens.

The test columns were stored in the testing room, the temperature of which during the summer months was about the same as that of the outside air, and varied during the winter months from 5 to 25° C.

The auxiliary test specimens, consisting of 8 by 16 inch concrete cylinders and 2-inch mortar cubes made from the material of the coverings as placed, were arranged in tiers and stored near the test columns. In some laboratory tests with mortar, the briquets and cubes were stored for periods in damp closet or water as stated in Section V (Table 28, Appendix D, p. 356).

V. AUXILIARY TESTS OF MATERIALS

The results of the auxiliary tests give information on the physical, chemical, and thermal properties of the materials used in the columns and coverings. Where generally accepted standards in the form of specifications exist some comparable measure of quality is thereby afforded. For the majority of the materials no general specifications have as yet been developed, and their representative character must be determined by the extent of their use and the methods employed to obtain material of average quality.

The large number of tests of concrete, mortar, and plaster give important information on their properties and variability as made under conditions approximating those obtaining in building construction.

1. TESTS OF STRUCTURAL, BAR, AND WIRE STEEL

Results of tension tests are given in Tables 5 and 6 (Appendix D).[1] The specimens of structural steel, about 7 inches long and ⅜ to ½ inch wide, were cut before test from the upper, enlarged portion of the column section by drilling and sawing. They were finished to uniform width over a gage length of 2 inches. Tests of bar steel were made on the full sections of the bars used except as noted in Table 5 (p. 336).

Specimens for hardness tests were taken where tension specimens were difficult to obtain, and results of tests are given in Table 7 (p. 338).

Chemical analyses of structural and rivet steel are given in Table 8 (p. 339).

The column steel was of the grade generally used for structural purposes. Of the 77 structural-steel columns used in the tests, 36 were donated by the manufacturers and 22 were bought from the same sources, with no specifications as to quality of material. The steel for the remaining columns, which were purchased at a later date, was specified to conform with the specifications for structural steel for buildings of the American Society for Testing Materials. Correspondence with the manufacturers indicates that all of the steel was made by the open-hearth process and that,

[1] The tabular matter for this section is placed in Appendix D (p. 334-364).

with the exception above noted, the specifications followed were manufacturer's standard specifications for steel for railway bridges or for medium open-hearth steel, tensile strength 55 000 to 70 000 pounds per square inch. A few of the test results are above or below the specification limits by 10 per cent or less, a variation that may be allowable, considering that most of the specimens secured were smaller than the standard size.

The metal for the reinforcing bars was specified to conform with specifications for billet-steel concrete reinforcing bars of the American Society for Testing Materials, structural grade. The spiral hooping was of hard-drawn wire of high tensile strength (87 400 pounds per square inch).

2. TESTS OF CAST IRON

Results of transverse and tension tests are given in Table 9. The specimens of the horizontally cast columns were cut from the projecting flanges in the upper 3 feet or head of the column. Those representative of the metal in the vertically cast-iron pipe columns 10A, 114, and 115 were cut from a duplicate column. Results of chemical analyses of the iron in this column are given in Table 10 (p. 339).

The iron for the horizontally cast columns was of gray foundry pig with admixture of machinery casting scrap. The mixture used for the cast-iron pipe column and caps was the same as that regularly used in the manufacture of cast-iron water pipe. The following analyses of the mix were furnished by the makers:

	Horizontally cast columns	Vertically cast—			Horizontally cast columns	Vertically cast—	
		Columns	Caps			Columns	Caps
Silicon	2.04	1.60	1.70	Phosphorus	.714	.....	.....
Manganese	.45	.34	.32	Sulphur	.108	.081	.060
Combined carbon	.72	.....	.....				

For the iron in the horizontally cast columns, the analyses and test results indicate conformity with accepted specifications for gray-iron castings of medium weight. For the vertically cast columns, tests of specimens cut from one end of the duplicate column gave transverse and tension values about equal to the specification limits and those of specimens cut from the other end gave values lower by about 15 per cent.

3. TESTS OF PORTLAND CEMENT

Tests of the Portland cement used in the columns and coverings were made by the Washington and Pittsburgh laboratories of the Bureau of Standards and by the R. W. Hunt Co. The results are given in Tables 11 to 13 (Appendix D). Sample Nos. 1 to 5 and B–1 and B–2, Table 11; H–1 to H–6, Table 12; and H–1 to H–3, Table 13, were all from the mill shipment received in April, 1916. Sample Nos. H–7 and H–8, Table 12, were of a later purchase of the same brand as the original mill shipment. Sample No. 12, Table 11, was of the Portland cement used in filling the pipe column of test No. 12. All samples were individual sack samples taken from the portions of the shipment used at the given time.

The time of setting was determined with the Gillmore needles in all tests. Other details of the test conformed with the specifications published by the American Society for Testing Materials, 1916. As based on average results, the tests indicate conformity with the given specifications within the pertaining limits and tolerances, excepting the test of sample No. 12. The latter gave low results in points of fineness and seven-day tensile strength.

4. TESTS OF SAND

The chemical and physical properties of the concrete and finer sands are given in Tables 14 to 17 (pp. 342–344).

The specific gravity was determined by measuring the volume of benzene displaced by a given dry weight of sand in a Le Chatelier apparatus.

The weight per cubic foot is that of the loose dry sand poured into the measure without shaking or bumping and leveled off even with its top.

The weight of the dry contents of a cubic foot of the sand as used on the work was somewhat less than that given in the tables due to the moisture contained which caused it to assume greater bulk than in the dry condition, the difference being about 10 per cent with coarse sand of 3 per cent moisture content. With the finer sands the decrease in weight was about 25 per cent as caused by a moisture content of 4 per cent.

The percentage of computed voids was taken equal to the following relation of values:

$$100\left(\frac{\text{Weight per cubic foot (dry)}}{1-\text{specific grav.}\times\text{weight of cubic foot of water}}\right).$$

The granular analysis gives percentages, by weight, passing the given sieve openings.

5. TESTS OF COARSE CONCRETE AGGREGATES

Chemical analyses and physical properties of the broken stone, gravel, and cinder used for concrete aggregate are given in Tables 18 and 19 (p. 344).

The physical properties were determined according to the methods given for tests of sand, except that apparent specific gravity was determined in water in a graduated vessel on the saturated aggregate and was taken as the ratio of the dry weight to that of the water displaced.

6. SOURCE AND CLASSIFICATION OF CONCRETE AGGREGATES

The sands and coarse aggregates used for concrete were analyzed for mineral composition and their chief constituents are given in Table 20 (p. 345). Further information on their mineral composition, geological origin, geographic location, and the method of preparing them for commercial use are given in the following descriptions:

(*a*) FOX RIVER SAND

This sand was obtained from a sand and gravel deposit about 3 miles east of Elgin, Ill. After crushing of the oversized gravel the whole is washed and screened to size. The grade is known in the Chicago market as coarse "torpedo" sand, the general maximum size of grains being ¼ of an inch. It is clean and very sharp, only 15 per cent of the grains being rounded.

The sand is of glacial origin, and its principal mineral constituents are calcite and dolomite, 44 per cent; quartz and chert, 39 per cent. Ferromagnesians, chiefly biotite, pyroxene, and hornblende, constitute 4 per cent. No free clay matter was present, although about 2 per cent of hydrous aluminum silicates were contained in the calcite and dolomite.

(*b*) JOLIET SAND

This is a clean, sharp, glacial sand obtained from a sand and gravel deposit at Rockdale. Ill., the method of preparation and

grading being the same as for Fox River sand. The chief mineral constituents are calcite and dolomite, 47 per cent; quartz, 42 per cent; feldspar, chiefly orthoclase ($KAlSi_3O_8$), 7 per cent; ferromagnesians, chiefly as biotite, mica, and hornblende, 3 per cent.

(*c*) MERAMEC RIVER SAND

This sand is obtained incidentally to screening and washing of gravel dredged from the bed or banks of the Meramec River at Drake, Mo. It consists almost wholly of quartz and chert grains formed from sandy chert by rolling in the stream bed. A small amount of calcite (1 per cent) is present as a coating on some of the grains. The quartz grains are fully rounded, while the chert is partly rounded or subangular.

(*d*) LONG ISLAND SAND

This sand was obtained from banks near Roslyn, Long Island, N. Y., and was screened and washed. It is a glacial sand formed from the crystalline rocks of New England. The principal constituents are quartz and feldspar, with minor amounts of ferromagnesians, slate, magnetite, and cinders. The cinders are not an original constituent of the sand. The sand is clean, sharp, and angular, the grains showing little rounding.

(*e*) PELEE ISLAND SAND

This sand was dredged by boat from the bottom of Lake Erie, at Fish Point, Pelee Island, Ontario. It is of glacial origin with sharp and angular grains that show almost no rounding. Of the mineral constituents, calcite and dolomite constitute 32 per cent; chert and quartz, 51 per cent. Clay is present to the extent of 4 per cent, of which about one-half is free clay matter and the rest shale.

(*f*) PLUM ISLAND SAND

This sand was dug on a beach at Plum Island on the Massachusetts coast and was not screened or washed. It is of glacial origin although much modified by ocean waves and currents. About 50 per cent of the grains have been worn to a subangular condition and about 3 per cent are well rounded. It is fine-grained and contains no free clay material, the mineral composition being otherwise closely identical with that of the Long Island sand.

(*g*) CHICAGO LIMESTONE

The stone, which was crushed to nominal ¾-inch size was quarried at Gary, Ind., from a ledge of the Niagara formation. It is a true dolomite (calcium magnesium carbonate), carrying a moderate amount of clayey impurities.

(*h*) JOLIET GRAVEL

This is obtained from the same source as Joliet sand. It is a glacial gravel, consisting largely of limestone pebbles, of which 20 per cent are rounded and 60 per cent subangular or partly rounded. Mineralogically the composition is 86 per cent dolomite, containing about 5 per cent clayey impurities; 5 per cent quartz in the form of sandstone and quartzite; 2 per cent quartz, 5 per cent feldspar, and 2 per cent ferromagnesian silicates as pebbles of granite, gabbro, and other basic igneous rocks.

(*i*) MERAMEC RIVER GRAVEL

The pebbles consist almost entirely of chert, an amorphous form of silica containing a variable amount of water in chemical combination. In the sample examined, 85 per cent of the pebbles show no trace of an earlier structure, 4 per cent are sandstone replacements, 9 per cent are limestone replacements, and 2 per cent have once been shale. The pebbles are colored brown with limonite ($2Fe_2O_3 \cdot 3H_2O$).

(*j*) NEW YORK TRAP ROCK

This was quarried at Haverstraw, N. Y., from the northern end of the sill of diabase which extends along the western shore of the Hudson River from this point to Staten Island, forming the Palisades of the Hudson. It is an intrusive igneous rock of finely crystalline texture. Of the minerals present, feldspar, chiefly plagioclase ($CaAl_2Si_2O_8$), with some albite feldspar ($NaAlSi_3O_8$), constitute 71 per cent; ferromagnesians as augite, 19 per cent, and as olivine, 7 per cent. In addition to 2 per cent of magnetite (Fe_3O_4), there are present scattered crystals of quartz (SiO_2) and apatite [$Ca_5F(PO_4)_3$].

(*k*) ROCKPORT GRANITE

This stone was quarried at Rockport, Mass., and is sometimes known as Cape Ann granite. It is an intrusive igneous rock that crystallized under great pressure from a fused condition.

The mineral composition is 59 per cent feldspar, chiefly orthoclase ($KAlSi_3O_8$) with some albite ($NaAlSi_3O_8$); 35 per cent quartz; 5 per cent ferromagnesian silicate, as hornblende [$(Fe, Mg)SiO_3$].

The feldspars show slight alteration to kaolin and chlorite. The quartz contains many tiny cavaties filled with gases and water occluded when the rock was formed.

(*l*) CLEVELAND SANDSTONE

The stone was quarried at Amherst, Ohio, and geologically is known as Berea sandstone or Berea grit. It is pure sandstone, 98 per cent consisting of subangular grains of quartz.

(*m*) HARD-COAL CINDERS

This is an anthracite cinder representative of the product used for cinder concrete in New York, N. Y. It consists largely of a porous, fused mass of basic silicates, apparently high in lime and magnesia and low in iron. There is present about 10 per cent of unburned coal and 5 per cent unfused ash.

7. TESTS OF CONCRETE

The quality of the concrete secured is thought to represent that of field-placed concrete as nearly as it can be conveniently duplicated under laboratory conditions. The method used for proportioning the dry materials was more accurate and the time of mixing longer than now generally obtain on construction work. The range of consistency resulting from the method employed for mixing the concrete is deemed to be representative of current field conditions. Where concrete is properly placed in large building members its properties are probably more uniform than as given by tests of the relatively smaller cylinders, being for a given mixture and consistency, more nearly the average values obtained in the tests.

(*a*) TEST SPECIMENS

Four cylinders of 8-inch diameter and 16-inch length were molded of the concrete mixed for each concrete covering, filling, or column. Cylinders were also taken of the concrete in the head protections of some of the columns where the method of mixing had been modified so as to give information on the effect of a lower water content and of a longer mixing period.

The molds were of cast iron with machined base plates. After the concrete had set and before removal of the molds, the cylinders were capped with a plastic mixture of Portland cement and calcined gypsum applied even with the top of the molds by means of a finished iron plate which remained in place until the mixture had set. In molding, care was taken to obtain the same consistency of concrete in the cylinders as in the column. Except for spading with a thin tool near the surface, the concrete was not puddled or tamped in the mold.

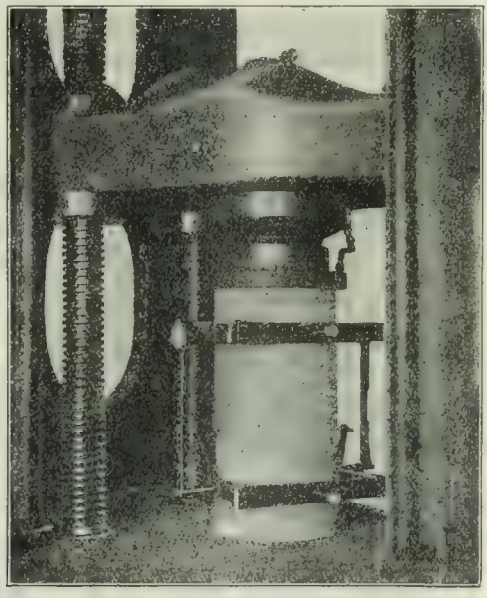


FIG. 14.—*Concrete cylinder after test*

(*b*) PER CENT WATER IN CONCRETE MIXTURE

The water introduced in mixing was measured at the mixer, and, for all but the first 15 columns covered, that present in the aggregates was determined from samples taken on the day the concrete was mixed. The dry weight in pounds per cubic foot of the aggregates as used was determined in a number of separate tests with each aggregate in which the same methods of filling the measures were used as were employed in proportioning materials for the concrete. The net weight per bag of Portland cement was found to be very nearly 94 pounds. The water content of the

mixture is the sum of the water introduced, and that contained in the aggregates and in the tables and diagrams is expressed as a percentage of the total dry weight.

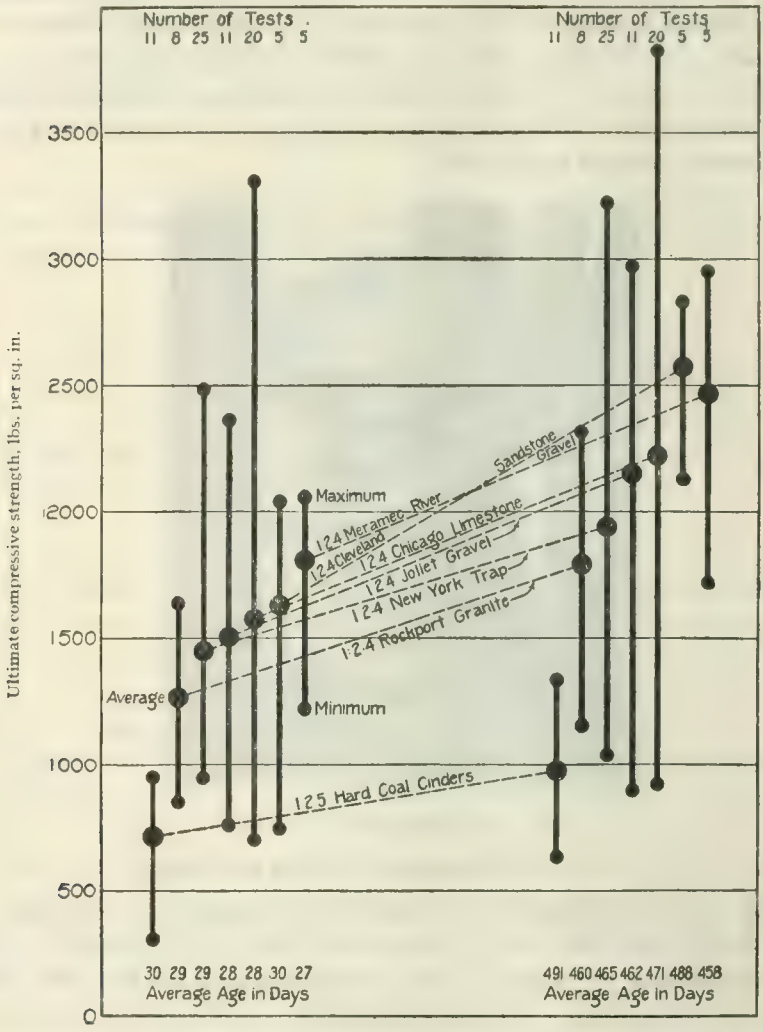


FIG. 15.—*Average and range of compressive strength of 1:2:4 and 1:2:5 concrete*

While the above method gives a fair comparison in point of consistency of concrete made from the same aggregate, the comparison between those of different aggregates is less direct, due to differences in specific gravity and absorption.

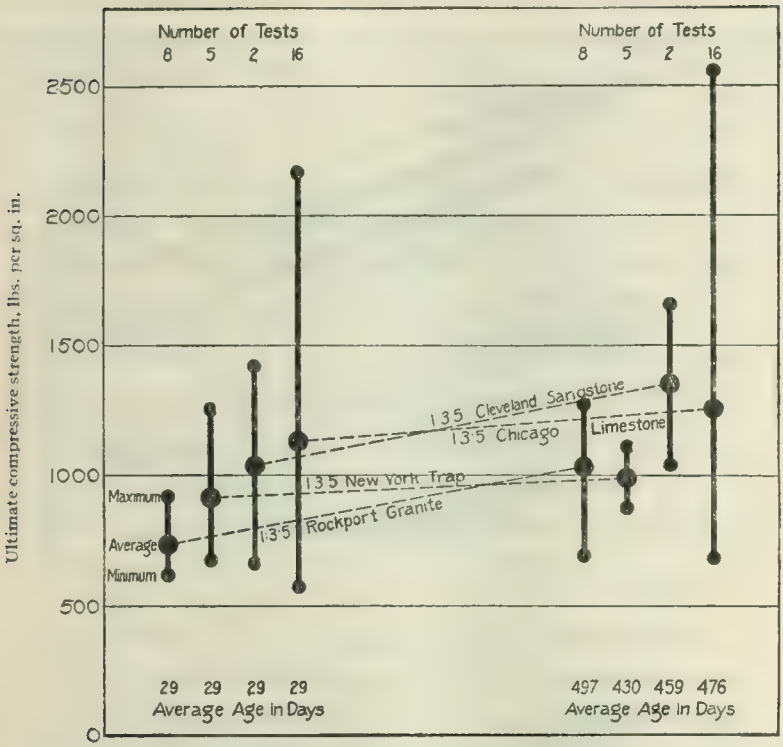


FIG. 16.—*Average and range of compressive strength of 1:3:5 concrete*

(*c*) **TESTING OF CONCRETE CYLINDERS**

Two cylinders of each set were tested in compression at the age of four weeks and two at the time the corresponding column was tested. In the case of the latter, the compressive load was applied by increments of about 10 000 pounds, and the deformation measured over a 10-inch gage length by means of an Ames dial gage mounted in a pivoted frame (Fig. 14). The results of tests on concrete mixed for the protections and columns are given in Table 21 (p. 346) and those of the concrete in some of the head protections in Table 22 (p. 350).

(*d*) **COMPRESSIVE STRENGTH OF CONCRETE**

The diagrams in Figs. 15 and 16 give the average and range of compressive strength of the cylinders taken from the concrete mixed for the columns, their covering and filling. In Figs. 17 and 18 the general effect of consistency on compressive strength is

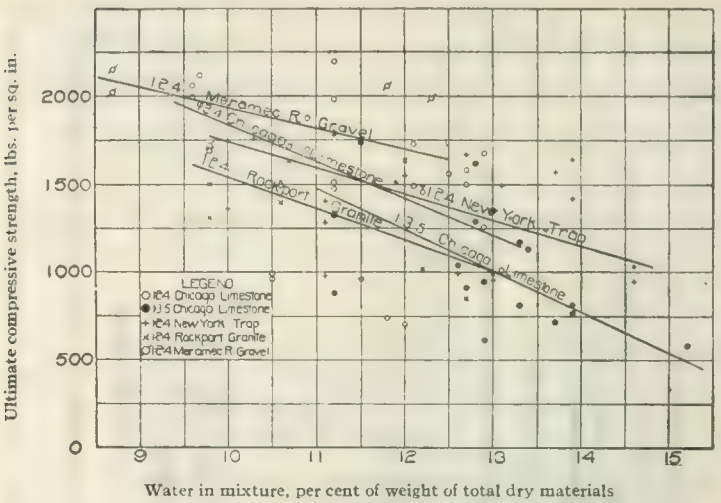


FIG. 17.—*Effect of consistency on compressive strength of concrete. Average age, 28 days*

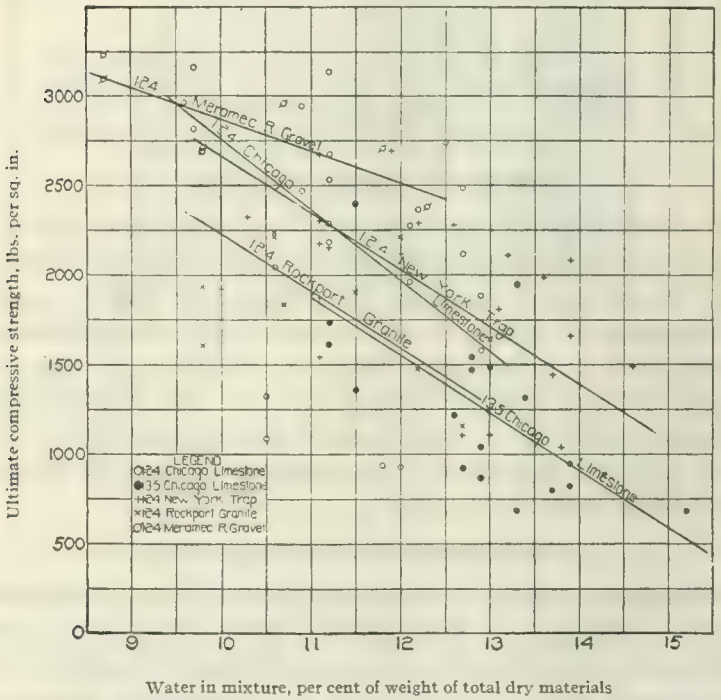


FIG. 18.—*Effect of consistency on compressive strength of concrete. Average age, 490 days*

shown. Herein are also included the tests (Table 22) made to study the effect of a lower water content. Fig. 19 gives a comparison of the strength of concrete mixed for 1 and for 2 minutes.

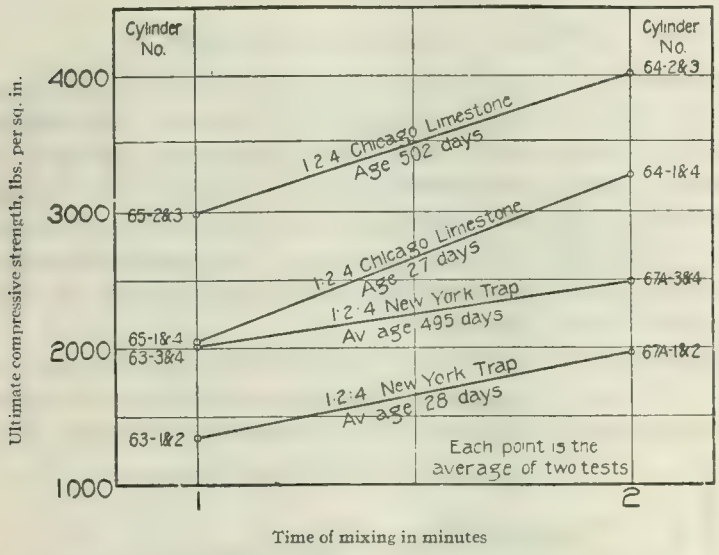


FIG. 19.—*Effect of time of mixing on compressive strength of concrete*

The mixtures having a water content of 9 to 11 per cent were of quaking or mushy consistency, the drier ones being discharged from the mixer with considerable difficulty. The higher water percentages (12 to 15) represent varying degrees of fluid consistency. The restricted space in which some of the concrete was placed made fluid or semifluid consistencies necessary in order to secure proper placement, although beyond the point where water collects above the concrete after discharge little is gained in this particular by the excessive use of water. For a given consistency the cinder and, to less extent, the sandstone, required a larger amount of water than the other aggregates, due to greater absorption.

For the concrete in the columns and coverings (Table 21) the amount of water introduced was determined by the man in charge of the mixing and placing of concrete for the given column, the water for the first batch being added by increments until the desired consistency was attained. For the concrete in the head

protections (Table 22) the amount of mixing water necessary for any desired total moisture content was generally predetermined, knowing the dry weight of the aggregates employed and their moisture content at the given time.

(e) MODULUS OF ELASTICITY OF CONCRETE

A comparison of values for 450, 650, and 850 pounds per square inch is given in Fig. 20 as obtained from Table 21. The modulus of elasticity is taken as the ratio of the given unit stress to the corresponding total unit deformation. In Fig. 21 the variation with ultimate compressive strength is shown, and in Fig. 22 the effect of water content of the mixture. The plots are based on values of modulus of elasticity given in Tables 21 and 22 for stress of 650 pounds per square inch.

In all tests the age of the concrete was over one year and the values of modulus of elasticity found are on the average higher than those generally taken to obtain for concrete at lower age.

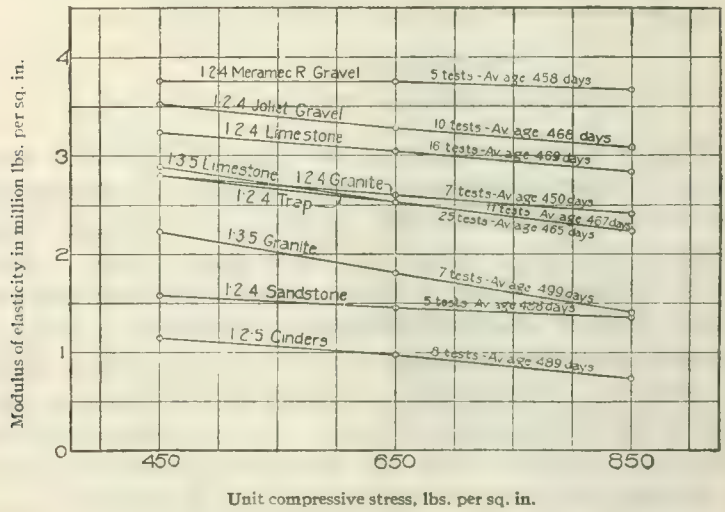


FIG. 20.—*Modulus of elasticity of concrete at 450 to 850 lbs. per sq. in.*

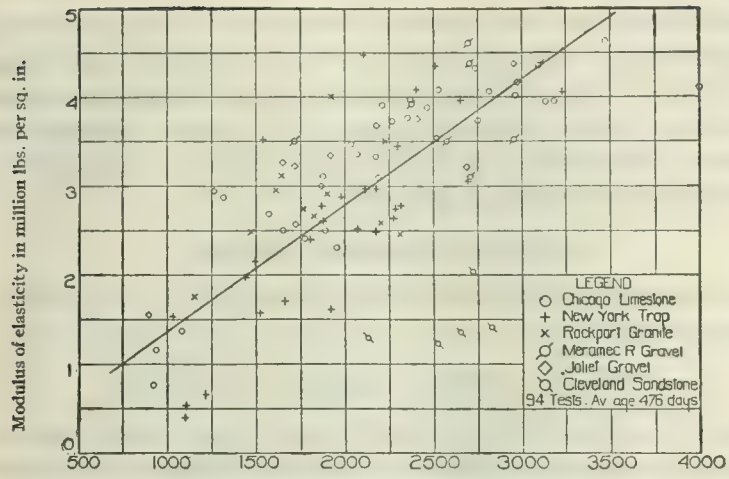


FIG. 21.—*Modulus of elasticity of 1:2:4 concrete at 650 lbs. per sq. in. Variation with ultimate compressive strength*

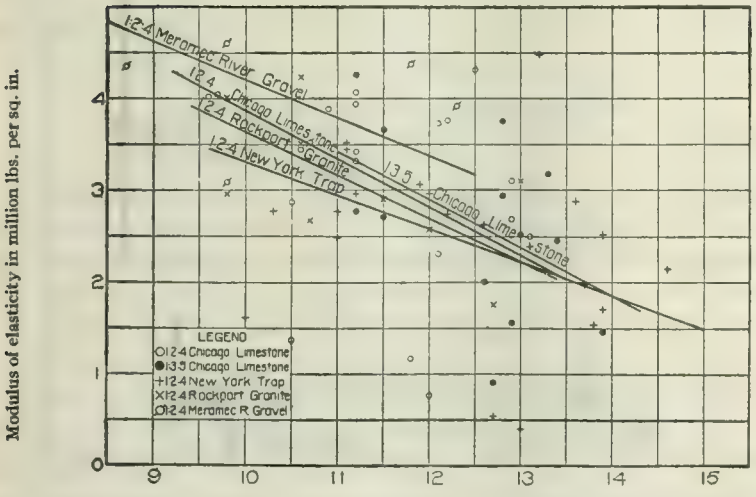


FIG. 22.—*Effect of consistency on modulus of elasticity of concrete at 650 lbs. per sq. in.*

8. TESTS OF LIME

Chemical analyses and physical properties of quicklime and hydrated lime are given in Table 23. The analyses disclosed a high calcium lime. The quicklime passed standard specifications but the hydrated product failed in point of CO^2 content, fineness, and soundness. The latter was used only as a minor component in the Portland-cement plaster protections.

9. TESTS OF CALCINED GYPSUM

Analyses and tests of eastern and western gypsum are given in Table 24 (p. 351).

10. TESTS OF MORTAR, PLASTER, AND FILLING

Results of compression tests on the mortar and plaster used in the coverings are given in Tables 25 to 27 and a further comparison of average and range of compressive strengths is given in Fig. 23 The specimens for these tests were taken of the materials as mixed by the workmen. They were stored in air without artificial drying or curing.

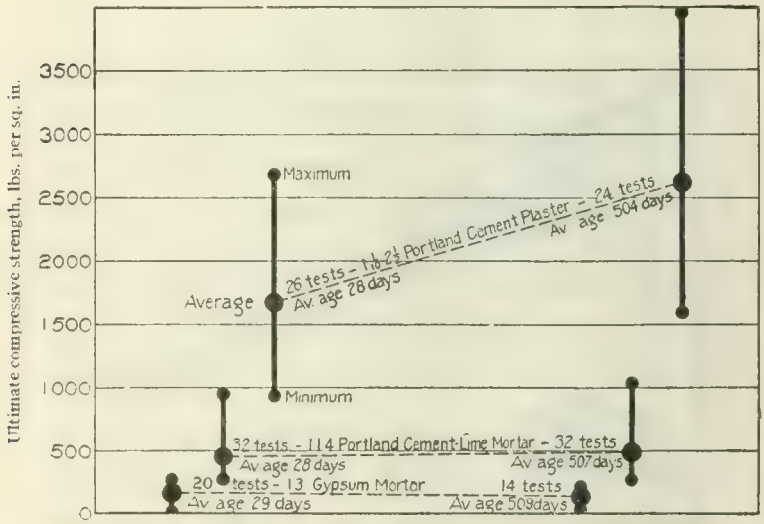


FIG. 23.—*Average and range of compressive strength of mortar and plaster*

The results of a series of mortar tests made by the Pittsburgh laboratory of the Bureau of Standards are given in Table 28 (p. 356). The same materials, proportions, and average percentage of water were used as for the mortar and plaster of the column coverings.

Table 29 (p. 354) gives results of compression tests on cylinders molded of the mixture with which gypsum-block protections were filled.

11. TESTS OF HOLLOW CLAY TILE

(*a*) CLASSIFICATION AND DESCRIPTION

The characteristics of the hollow tile employed as column covering are given in Table 30 (p. 357).

The straight partition tiles, Nos. A to E, were nonporous; that is, burnt without sawdust or other filling. The curved tiles, Nos. F, G, and H, in point of method of manufacture, were porous.

(*b*) POROSITY AND ABSORPTION

The porosity of the burnt clay (Table 31) was obtained by drying samples of tile to constant weight at 112° C and immersing in hot water under vacuum of 24 inches of mercury for about four hours, the porosity being based on the ratio of volume of water absorbed to that of the test specimen. The weight of water absorbed is also expressed in Table 31 (p. 358) as a percentage of the dry weight.

(*c*) COMPRESSIVE AND TRANSVERSE STRENGTH

Table 31 gives results of compression tests of all types of hollow tile used in the column coverings. The curved tile was tested on end and the partition tile was tested both on end and on edge. By the latter method the load was sustained by the outer shells only and the average maximum unit loads developed are in all cases lower than for the tests made with the tile on end.

The transverse tests (Table 32) were made with center load and supports 10 inches apart. Failure generally occurred on inclined planes extending from points on the lower surface upward toward the loading line. In calculating the maximum outer fiber stress and horizontal shear, the moment of inertia and other properties of the section were obtained, using the measured dimensions of each test specimen.

(*d*) TEMPERATURES OF VITRIFICATION AND FUSION

In determining temperatures of vitrification a number of pieces of each kind of tile were dried and weighed as for the porosity

tests, and heated to 800° C in a muffle furnace, after which samples were withdrawn at temperature intervals of 30° C, allowed to cool slowly, and tested for porosity. The temperature of vitrification was taken as the point or region of minimum porosity.

In preparing samples for the fusion test, fragments of a number of tiles from the same clay were broken up and a sample obtained by quartering. This was ground and again sampled, the final sample being ground to pass through an 80-mesh sieve and made into cones ⅞ of an inch high. These were fired in a small pot furnace between standard cones, the temperature of the furnace being determined with platinum thermocouples. Softening was indicated when the tip of the test cone went over. Fusion was taken as occurring at the stage when the tip of the cone reached the level of its base.

The temperatures of vitrification and fusion determined for the hollow tile and brick are given in Table 33 (p. 361).

12. TESTS OF BRICK

Results of tests giving the porosity, absorption, compressive and transverse strength of the brick used in the brick protections are given in Tables 34 to 36 (p. 361–362).

The brick was made in the Chicago district of calcareous surface clay and burnt to medium hardness, the color being generally light red. The average dimensions in inches were 2¼ by 3¾ by 8.

13. TESTS OF GYPSUM BLOCK

Results of porosity, compression, and transverse tests of gypsum block are given in Tables 37 to 39 (p. 363–364).

The porosity tests were made on pieces of block dried for 24 hours at 60° C and saturated in kerosene under vacuum, the method being in other respects the same as given above for clay tile.

In calculating modulus of rupture the outside dimensions of the block were used, the scoring and imprints being neglected.

14. TESTS OF GYPSUM WALL BOARD

Results of transverse tests on gypsum wall board, in the dry condition and after saturation in water, are given in Table 40. The strength of the board was limited by the tensile strength of the paper facing, the latter developing greater resistance when the strain was parallel with the grain of the paper.

VI. DESCRIPTION OF FURNACE AND RELATED EQUIPMENT

The furnace and accessory equipment used in the tests were specially designed for the purpose and consist of a carriage and traveling crane for handling the test columns, a furnace in which the columns are subjected to fire, a ram with restraining frame for applying load, and hydrant with means for appplying water in the fire and water tests.

1. BUILDING

The tests were made at Underwriters' Laboratories, Chicago, Ill., in a building designed for work of this character, a partial plan of which is given in Fig. 24 and a sectional elevation through the center in Fig. 25.

The central portion of the building, wherein the furnace for testing columns is located, is in one story about 37 feet high, an intermediate operating floor being located in the outer bays. Sliding skylights in the roof of the building above the furnace provide means for ventilation. The operator's station on the upper floor is inclosed within fire-resistive partitions, with openings protected by wired glass.

2. APPARATUS FOR HANDLING COLUMNS

A carriage traveling on the lower flange of the **I** beams placed above the storage bays for the columns is used for transferring the latter to the traveling crane. The column to be moved is lifted on threaded rods passing through the top plate of the column and plates supported on a truck contained within the carriage. This truck carries the column from the traveling crane to its location in the furnace, on **I** beams (*a*, Fig. 25) spanning the space above the furnace (Fig. 24).

The column is finally attached to the head of the loading ram and lowered into position.

3. LOADING APPARATUS

(*a*) LOADING RAM

The special hydropneumatic ram used for loading the columns was designed to maintain a constant load during the test and to develop characteristic deformation at the point of failure.

The main ram is 36 inches in diameter and is connected with a smaller lifting ram supported on its upper head. The operating

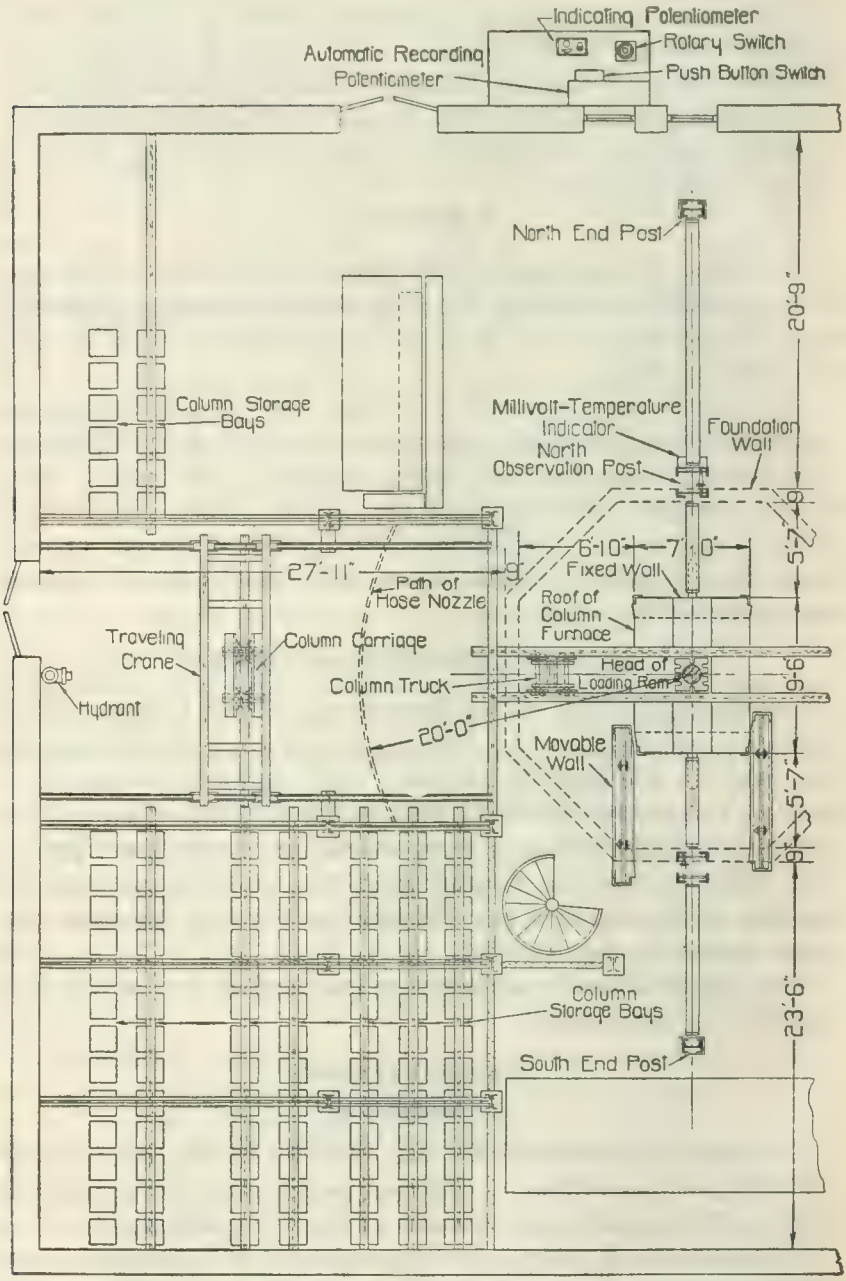


FIG. 24.—*Plan of testing room*

medium is water maintained at the required pressure by an air compressor which discharges into steel pressure tanks connected with the ram cylinder through a 6-inch main. A pump is also provided to maintain the water at any given level.

Provision is also made for applying load by pumping water directly into the main cylinder. The gate valve in the main leading to the air-pressure tanks is in this case closed and regulation of pressure obtained with a spring relief valve.

(*b*) CONTROLLING DEVICES

A gate valve is placed in the pipe connection between the pressure tanks and the main cylinder, and also an automatic cut-off valve which shuts off the pressure tanks and releases the pressure in the main cylinder after a predetermined downward movement of the plunger of the ram has taken place. For the present tests the valve was set to begin shutting off pressure after the plunger had gone down 2 inches, the valve releasing the pressure after an additional 1¼-inch downward movement. This caused maximum center deflection of the column at failure of about 15 inches. As a further precaution a pressure of not less than 400 pounds per square inch was maintained in the lifting cylinder.

The portion of the main cylinder chamber below the piston is connected with the outside air and also with a closed discharge chamber, with the latter through perforations in the cylinder lining. A depth of water is maintained in this part of the cylinder to act as a cushion in case the travel of the ram piston, following a sudden column failure, should extend to the end of the cylinder.

To obtain an approximate record of the load sustained by the column during test and during the period of failure, a steam-engine indicator was mounted on the control board (*s*, Fig. 27) with its pressure chamber connected with the main cylinder of the ram and the drum actuated by a cord attached to the ram plunger. The resulting card diagram indicated on axes at right angles, the pressures in the main cylinder and the corresponding positions of the ram plunger.

For convenience in operating, the valves for admitting and releasing pressure to all parts of the loading system are located at the control board (Fig. 27). Here are also located the load and pressure indicating gages and the automatic and manually oper-

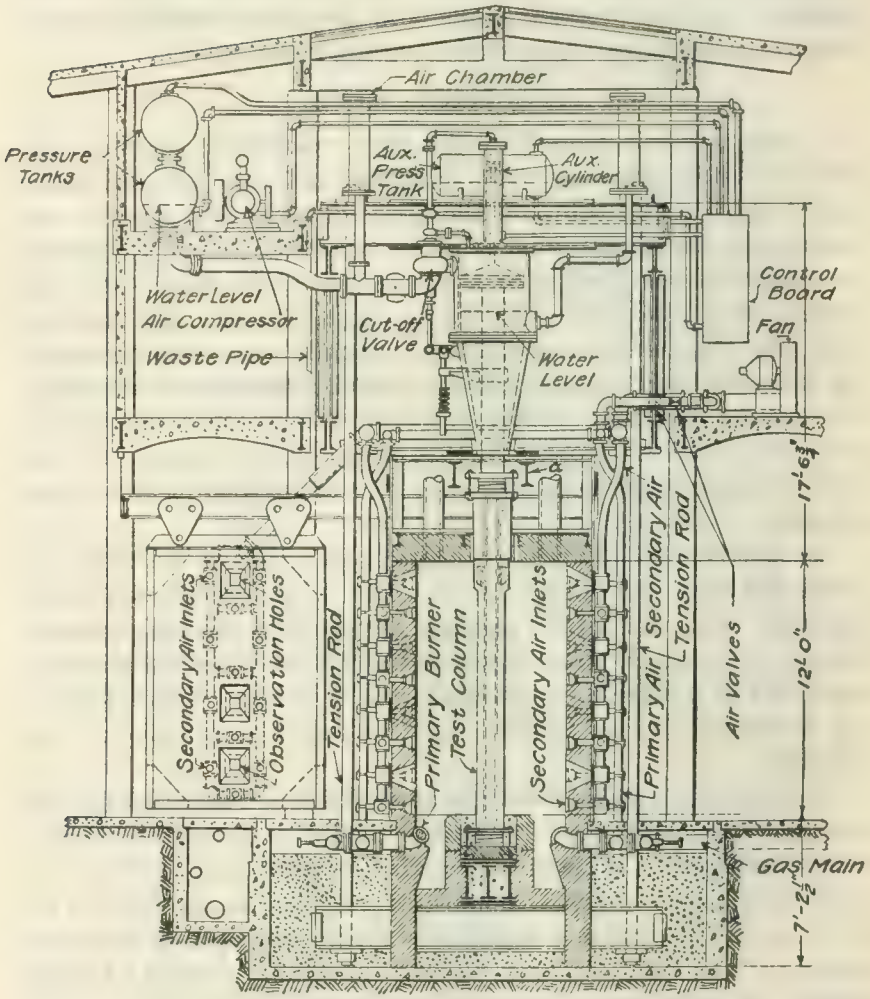


FIG. 25.—*Elevation of testing machine*

ated controls for the starting switches of the air compressor and water pump. The main features of the installation are shown in Fig. 25, minor piping and details being omitted.

(*c*) RESTRAINING FRAME

The ram is supported from the upper transverse members of the frame, the lower set being embedded in the foundation pit. Each set consists of two pairs of 24-inch I beams with cover plates. Two vertical tension bars on each side carry the reactions from the applied column loads. The lower end of the ram is supported laterally in an intermediate cross frame.

(*d*) BEARING DETAILS

The ends of the test column are bolted to rolled-steel plates 2 inches thick, which are in turn bolted to a cast-steel foundation plate at the bottom and to the lower flanged head of the ram at the top. Between these and the column plates are placed adjustable bearings consisting of skew disks, the middle two of which can be turned to obtain even bearing.

(*e*) CAPACITY AND CALIBRATION

The machine has a load capacity of 545 000 pounds and has been calibrated by the Bureau of Standards, using a comparison bar whose load-deformation relation was determined in the 2 300 000 pound Emery testing machine at Washington.

The accuracy of loading within the range used in this series of tests can generally be taken to be within 1 per cent of the applied load, with a possible extreme variation of 2 per cent. The errors incurred are due to variation in the ram friction and in the indication of the test gages used for determining the load. The latter were calibrated in a dead-weight gage tester before or after each test.

4. TESTING FURNACE

(*a*) COMBUSTION CHAMBER

The chamber is 7 feet square and 12 feet high, exclusive of two shallow pits to receive falling material and carry off water during the fire-stream tests. The sides are formed by two stationary brick walls and two movable brick walls suspended in structural-steel frames from overhead beams by trolleys (Fig. 25). These walls are pulled open by means of a motor-driven hoist. The top of the furnace consists of heavy fire-clay blocks supported

by steel I beams and hangers. Some of the blocks are removable to permit installation of the test column. Concrete made of Portland cement and crushed fire brick and reinforced with rods and wire mesh was later substituted for fire clay in making additional roof blocks. The bottom of the chamber is formed by the protection provided for the lower bearing plate and the supporting members of the restraining frame.

The products of combustion are carried off through four flues extending from the furnace through the roof of the building. The furnace walls are provided with mica-glazed observation holes, so arranged near the top, middle, and bottom that all parts of the test column can be observed.

A general view of the furnace with a preliminary test column in place is given in Fig. 26.

(*b*) BURNERS

The furnace is heated by means of four primary blast burners extending in at the corners at the bottom of the chamber, and are all arranged to discharge in an inclined direction upward and toward the adjacent corner. The burners are supplied with gas and air through tubes connecting with mains located in a shallow pit under the floor outside of the furnace. Gas is supplied to the furnace through a 6-inch pipe connected to the city service mains. Air is supplied by an electrically driven blower.

The primary burners are operated with a smaller proportion of air to gas than is required for full combustion. The additional air needed to complete combustion and distribute the fire is supplied by secondary air inlets extending through the walls of the furnace at regular intervals toward the top, each inlet being provided with a regulating valve (Fig. 25). The air is so directed as to avoid impingement on the test column, and to set up a whirling motion of the furnace gases as an aid in securing uniform temperature distribution.

(*c*) OPERATING DETAILS

Means for regulating the gas and air supplied to the burners and the draft from the furnace chamber are located on the main floor An emergency cut-off valve with electrical control is placed in the gas main leading to the furnace, with switches for operating it set at several points in the building.

A Hoskins direct-reading millivolt-temperature indicator with connections to four chromel-alumel furnace thermocouples is placed near the furnace for use in regulating its temperature.

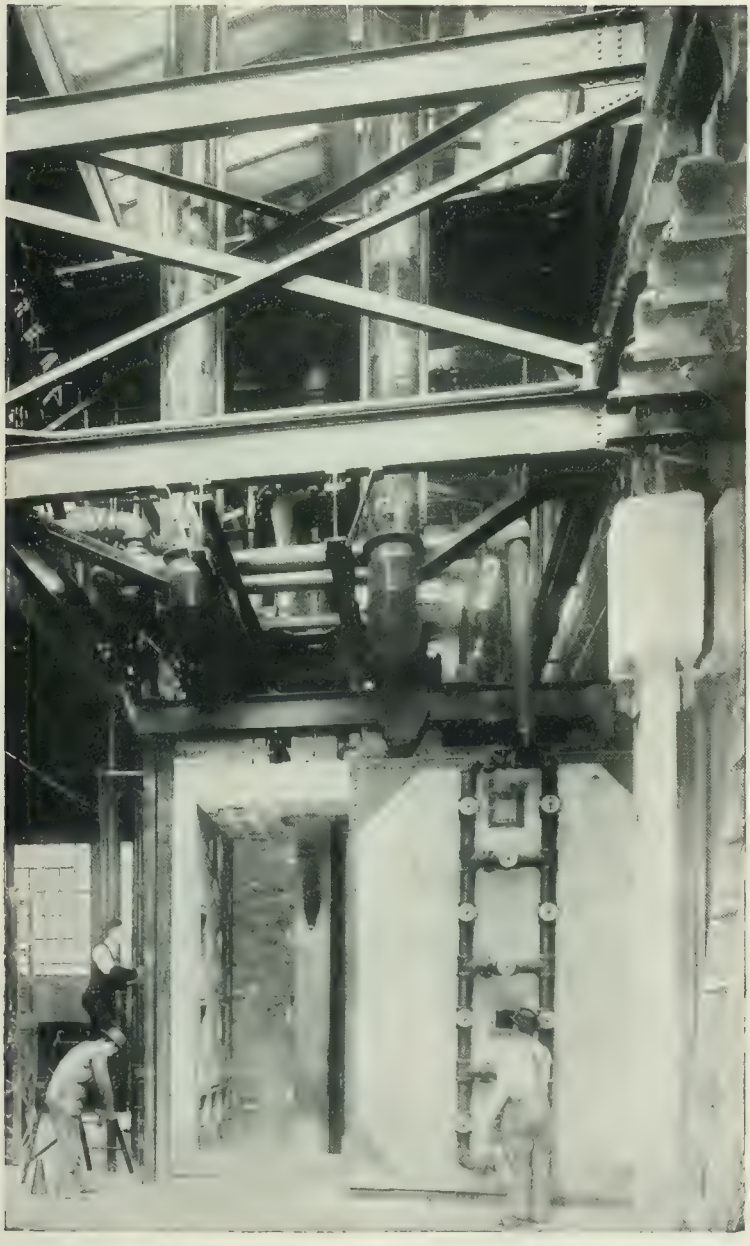


FIG. 26.—*General view of testing machine*

A—Gate to main cylinder.
B—Air outlet from auxiliary tank
C—Water outlet from auxiliary tank
D—Water outlet from main tanks
E—Air outlet from main tanks
F—Water pump switch.
G—Air compressor switch
H—Automatic pressure regulator
I—Water inlet to auxiliary cylinder.
J—Water outlet from auxiliary cylinder.
K—Air inlet to auxiliary tank
L—Water inlet to auxiliary tank
M—Water pump pressure
N—Water inlet to main tanks
O—Air inlet to main tanks.
P—Air pump pressure.
Q—Auxiliary cylinder pressure.
R—Main cylinder pressure.
S—Pressure indicator
T—Spring relief valve
U—Auxiliary tank pressure
W—Main tank pressure.

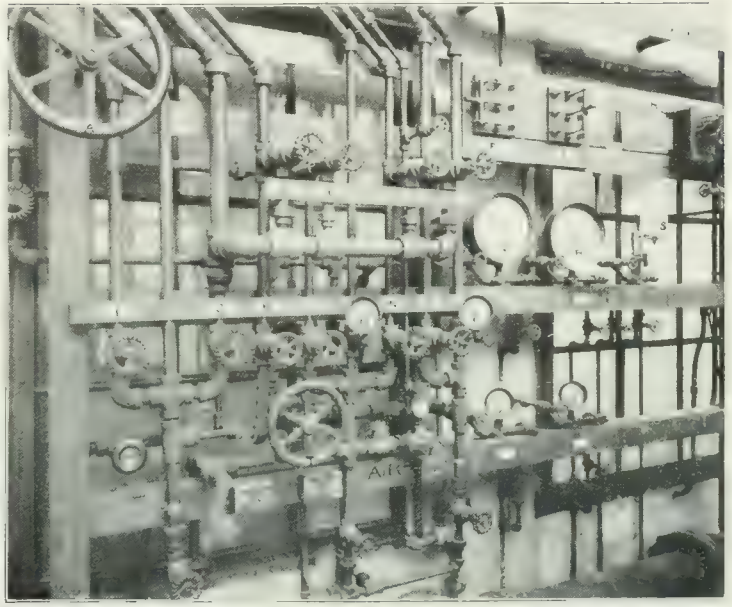
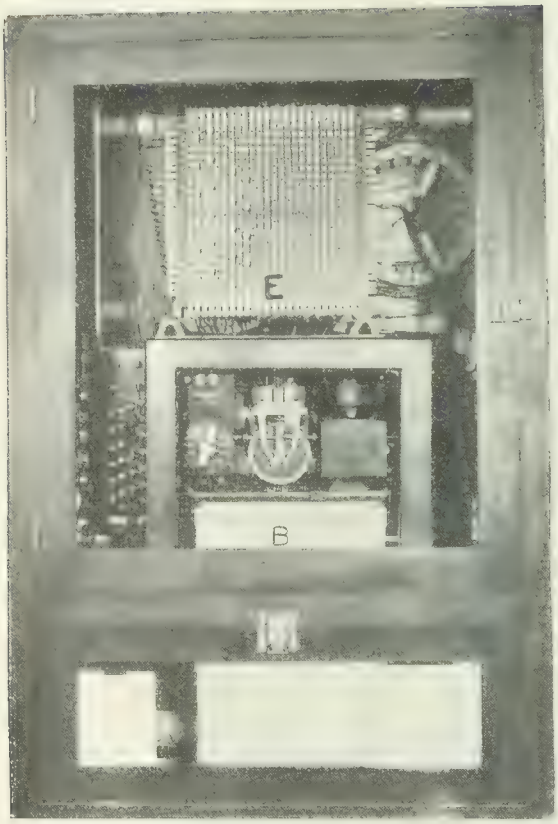


FIG. 27.—*Control board*

FIG. 28.—*Fire-stream apparatus*



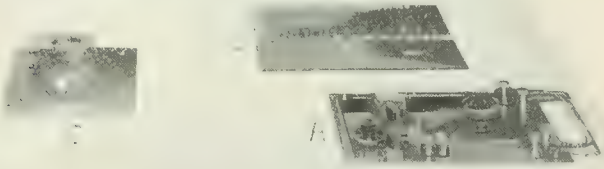


FIG. 20 — *Temperature measurement apparatus*

5. FIRE-STREAM APPARATUS

The pressure head for the hose stream used in the fire and water tests is supplied by air pressure in a tank of 4500 gallons capacity. The water is applied through a standard Underwriter play pipe with nozzle 1⅛ inches in diameter (Fig. 28). It is connected with the hydrant through about 20 feet of 2½-inch cotton, rubber-lined hose. A throttling valve is placed at the outlet to the hose and a pressure gage connected with the base of the nozzle by means of which the desired pressure during water application is maintained.

The water was applied to three sides of the column, the nozzle being moved back and forth on one side of the furnace and kept at a distance of about 20 feet from the center of the column (Fig. 24).

VII. TEMPERATURE MEASUREMENTS

All temperature measurements were made by the thermoelectric method, platinum-platinum rhodium being used in the thermocouples placed in the furnace and base metal in those on the test columns. The electromotive force was measured with indicating and recording potentiometers, and the corresponding temperature obtained from comparison calibrations with standard thermocouples whose temperature-emf relations were known.

The temperatures are expressed on the Centigrade scale. The Fahrenheit equivalents are given at the right of the time-temperature plots. A centigrade and Fahrenheit conversion table is given in Appendix F (p. 375).

1. INSTRUMENTS

A view of the instruments used is given in Fig. 29, wherein an indicating potentiometer or indicator is shown at *A*, a recording potentiometer at *B*, dial and push-button switches at *C* and *D*, respectively, and bus-bar distributing board at *E*.

(*a*) INDICATING POTENTIOMETER

The potentiometer consists essentially of a means for securing a known variable electromotive force and balancing it against the electromotive force to be measured.

A wiring diagram of a simple form of this instrument is shown in Fig. 30. Current from a battery flows through the resistance

ABCDE, of which the portion *BCD* is a slide wire. A scale is attached to the slider, giving the potential drop between points on the slide wire and the point *B* for a given value of the battery current. The latter is adjusted to the given value by opposing the electromotive force of a standard cell to the drop of potential over the resistance *AB* and varying the resistance *R* until the galvanometer included in the circuit shows no deflection. In a similar manner, an unknown electromotive force as that of a thermocouple, is balanced against the potential drop over some portion *BC* of the slide wire and its magnitude read from the attached scale.

Corrections for variation in the temperature of the cold end of the thermocouple require in general the addition of a small electromotive force to the observed one. With the potentiometer used

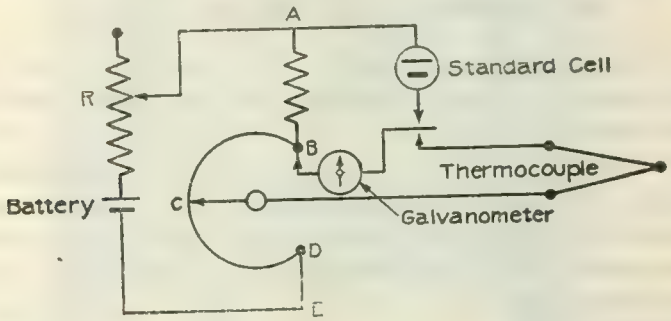


FIG. 30.—*Wiring diagram of indicating potentiometer*

in these tests, this could be accomplished mechanically at the time of making the observation by setting off the corresponding electromotive force on the cold-junction compensator, which has the same effect as moving the point *B* up the scale the same amount.

The potentiometer indicator used in these tests is equipped with two scales, one reading from 0 to 16 and the other from 0 to 80 millivolts. The accuracy of this particular instrument was found to be within one-fourth of 1 per cent of the total range at all points on the scale.

In the above method of measurement the indication is independent of the resistance of the thermocouple and lead wires, and no errors are introduced by variations in resistance caused by temperature changes in them.

(*b*) RECORDING POTENTIOMETER

The wiring diagram of this instrument is an exact equivalent of that of the indicating potentiometer (Fig. 30), the electromotive force of the thermocouple being automatically balanced and recorded. It has scales of the same range as the indicating instrument, and a 13-point selector switch provides connections for a maximum of 12 thermocouples. On the remaining switch point the battery current is automatically adjusted to the correct value. Records are made at intervals of one minute. Where the temperature differences between the different couples connected are not too large, records can be made every half minute.

(*c*) ACCESSORIES

The accessories consist of a bus-bar board that was used for connecting any desired set of thermocouples with the recording potentiometer, a 12-point double-pole, double-throw, push-button switch, by means of which the couples connected with the recorder could be read on the indicator, and a 24-point double-pole dial switch for connecting thermocouples directly with the indicator.

2. FURNACE TEMPERATURES

Because of the high temperatures attained in column tests of long duration it was deemed advisable to use rare metal (Pt–90 Pt 10 Rh) thermocouples for the measurement of furnace temperatures.

(*a*) LOCATION OF FURNACE THERMOCOUPLES

In general, four couples were used to measure furnace temperatures, placed two on each of two levels at 3 feet and 9 feet above the fireproofing about the base of the column (Fig. 31). The couples on the 3-foot level passed through the furnace walls near the northwest and southeast corners; those at the 9-foot level near the northeast and southwest corners. Laterally the junctions of the furnace pyrometers were placed 15 inches from the nearest point on the column. The above symmetrical distribution is such that the center of gravity of the couple locations is identical with that of the portion of the furnace chamber above the fireproofing line at the base of the column.

The curves on the time-temperature plots (Appendix B) corresponding to the respective furnace couples are designated as *L–NW*, *L–SE*, *U–NE*, and *U–SW*, the letter preceding the dash indicating the level (*L*, lower; *U*, upper) on which the corresponding couple was placed; those following it, the corner nearest to which the couple entered the furnace.

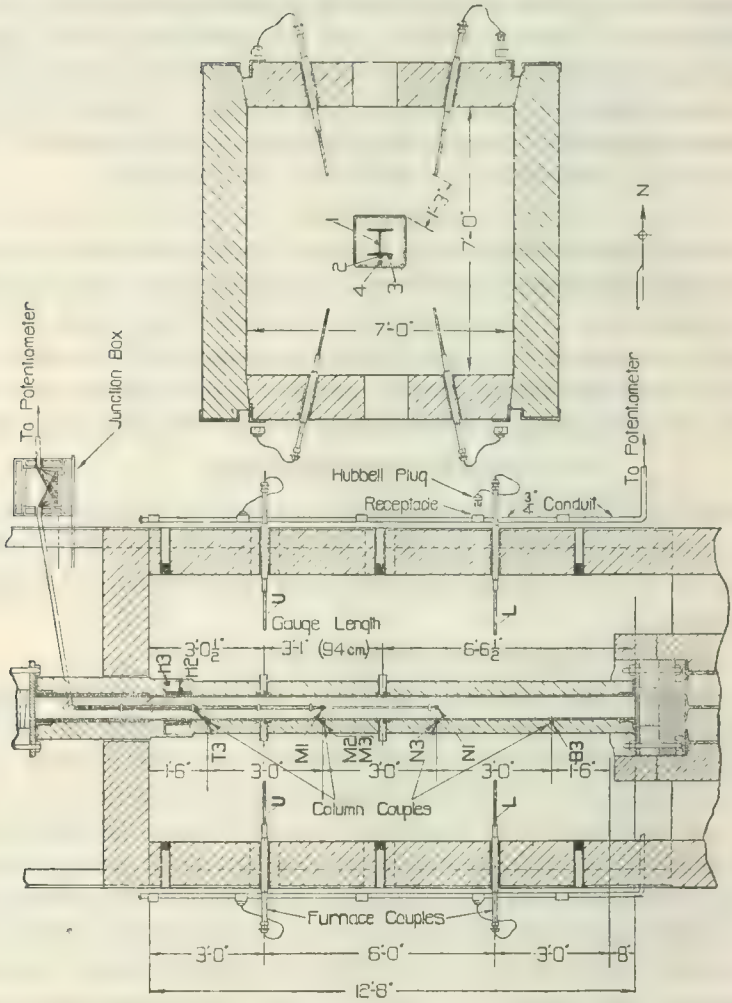


FIG. 31.—*Location of furnace and column thermocouples*

The same system of designation is extended to include the few cases where couples were used in locations other than those mentioned. An exception is found in test No. 20, in the case of curves $L4$ and $U4$. The corresponding couples measured furnace temperatures at points 1½ inches from the column face, 3 feet and 9 feet, respectively, above the fireproofing line. They were made of No. 16 iron and constantan wires, with the junctions exposed directly to the furnace atmosphere.

(*b*) THERMOCOUPLE MOUNTINGS

Details of the thermocouple mounting are shown in Fig. 32. The use of rare-metal couples necessitated porcelain protecting tubes, which were covered with woven-asbestos sleeving to prevent breakage from sudden temperature changes. The alundum tube, with its winding of asbestos cord covered the outer 2 feet of the porcelain tube and served as a protecting sleeve where the mounting passed through the furnace walls.

The porcelain protecting tubes differed somewhat in wall thickness, as furnished by three different makers. The maximum and the minimum wall thickness were $\frac{5}{32}$ and $\frac{1}{16}$ of an inch, respectively. About 85 per cent of the total number of tubes used had a wall thickness of very nearly ⅛ of an inch.

Practically all of the woven asbestos sleeving was bought in two lots, both of the same nominal size. The first lot, comprising approximately ¾ of the sleeving used, ran about 15 feet per pound, the second lot about 24 feet per pound, as unwound from the spool.

(*c*) CONNECTIONS TO INSTRUMENTS

Permanent leads of No. 18 copper fixture wire were installed in a conduit leading from the case housing the recording potentiometer to the base of the furnace wall, from whence branches extended up to the thermocouple points. Hubbell receptacles were provided at the couple outlets (Fig. 31), which received the Hubbell plugs terminating the short leads attached to the cold ends of the furnace couples (Fig. 32). At the instruments the leads led to the bus-bar board, where each of the four furnace couples was connected to three recorder points, and also through the push-button switch with the indicator.

The temperatures of the cold junctions in the heads of the furnace pyrometers were measured by thermocouples formed of one of the copper leads in each head and No. 18 constantan wires

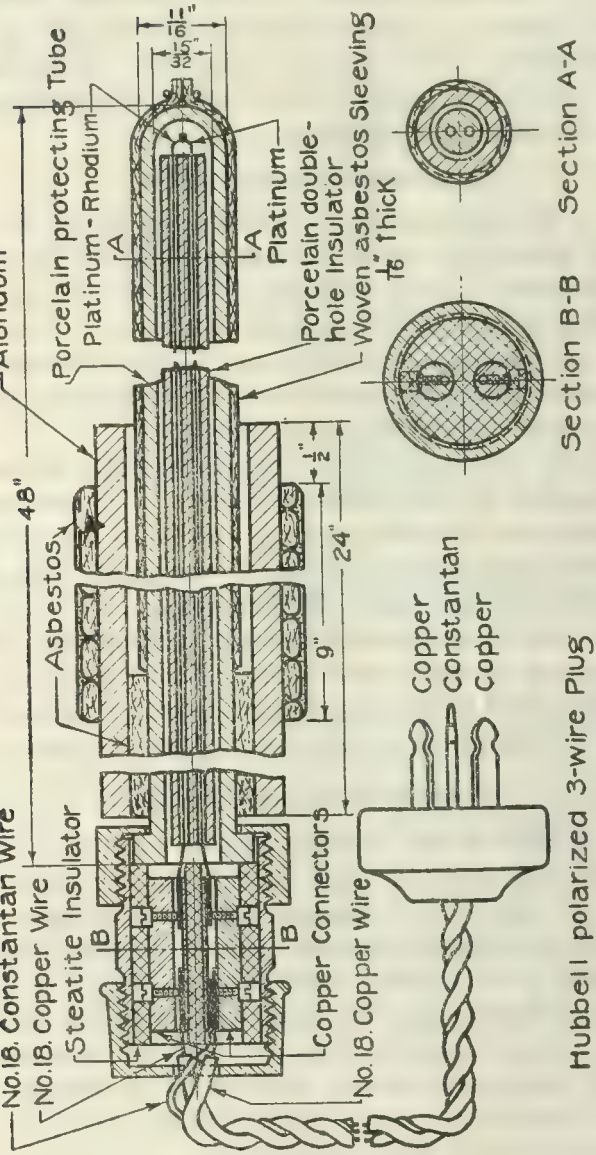


FIG. 32.—*Detail of furnace thermocouple mounting*

(Fig. 32), which were installed in the same conduit as the copper leads, all wires extending to the case housing the recorder, from whence copper wires led through the dial switch to the indicator.

3. COLUMN TEMPERATURES

The base-metal thermocouples used for measuring column temperatures were of No. 16 (B. & S. gage) iron, No. 16 and No. 18 constantan, and No. 18 nickel wires, in the combinations, iron-constantan and nickel-constantan, 90 per cent of the total number of couples used being of the iron-constantan combination. As supplied, a portion of the wire carried a thin waterproofed asbestos covering, while the remainder was bare. As used on the column all wire was covered with closely fitting asbestos sleeving.

(*a*) ATTACHMENT OF THERMOCOUPLES TO COLUMN

A preliminary test consisting in subjecting to heat in a gas-fired furnace two concrete-covered lengths of wrought pipe, with thermocouples placed on the metal and in the covering, indicated that the proposed method of measuring column temperatures gave, in general, reliable results. Two methods of attaching the couples to the pipe section were tried out and appreciable differences of indication found to be caused by the given variation in this detail. By the one method, which will be termed "peening" the ends of the individual couple wires were placed in closely-fitting holes drilled in the metal and secured in place by driving in the metal around the holes, using a slotted punch placed in turn over each wire. Couples attached by the other method, and referred to as "insulated," had fused or twisted junctions that were laid against a piece of mica about 0.01 inch thick placed between it and the metal.

Since it was apparent that the peened couples gave more nearly the true temperature of the metal, the couples on the columns prepared subsequent to the preliminary test were attached by peening, except that one or two insulated couples were added at certain important locations as a check on the adjacent peened couples.

The couples placed before the preliminary test was made were all insulated and include those under the concrete covering or filling of all partly protected columns, test Nos. 14 to 22, inclusive, and the couples of the concrete-protected column of test No. 28 and those on the vertical bars of the reinforced-concrete column of test No. 73.

To decrease the danger of injury to the peened junctions in placing the coverings and in testing the columns, the region immediately surrounding the point of attachment was covered to a thickness of about ⅛ of an inch with asbestos wool loosely placed under a shield of sheet iron 1 inch square and 0.015 inch thick. The couple wires, protected by the applied sleeving, were held firmly in place under a strap-iron clip screwed to the column near the junction. The same clip also held the shield over the asbestos wool in place. Similar clips, placed about 18 inches apart, held the couple wires on the column up to 1 foot below the top bearing, where they were all led out through a 1¼-inch pipe (Fig. 31).

The asbestos wool and shield were omitted over the insulated junctions, since it was thought that they would not be subjected to as large strains as the junctions that were peened.

As it was very difficult to attach couple junctions to the inside of hollow steel or cast-iron columns by the peening method, the junctions were laid against the metal and protected with asbestos wool, as described above.

In placing couples in the column coverings or in locations other than on the metal, suitable means were used for supporting them in place, the methods varying with the conditions presented by the different types of columns and coverings. In all cases the details were so arranged as to reduce to a minimum heat interchange between the couple junction and its support as well as between the junction and the couple wires. The latter was accomplished by placing a sufficient length of the wires immediately leading from the junction in a plane where, with the contemplated fire exposure, the same temperatures would normally obtain as at the junction.

(*b*) LOCATION OF COLUMN THERMOCOUPLES

In all tests, except in those of timber columns, the thermocouples were spaced vertically in four general locations, *B*, *N*, *M*, and *T*, 1 foot 6 inches, 4 feet 6 inches, 7 feet 6 inches, and 10 feet 6 inches, respectively, above the fireproofing line at the base of the column (Fig. 31). The lateral locations are numbered 1, 2, 3, 4, etc., and marked by small solid circles on the sectional location diagrams of the time-temperature plots in Appendix B. The letter and the number designating, respectively, the vertical and horizontal position, as combined in the curve designations, define completely the location of the couple.

Additional couples, designated on the plots (Appendix B) as $H2$ and $H3$, were placed on some of the structural-steel and cast-iron columns at the base and on the edge of the beam bracket near the top (Fig. 31).

In general the letter *i* following the numbers in the column couple designations (Appendix B) is to be interpreted as "*insulated*," referring to the method of attachment. Exceptions occur only in test Nos. 3 and 4 (Fig. 90) in the case of the couples attached on the inside of the column.

Letters indicating points of the compass are, in general, included in the designations of the couples that are not on the side of the column on which the general vertical distribution occurs. In the latter case, the couples are designated by the location letter and number only. Exceptions to the above are designations for the insulated couples in test No. 27 (Fig. 98), No. 47 (Fig. 101), and Nos. 62 and 63 (Fig. 126), which are located near the couple on the same level that has a direction letter in its designation.

The couples whose curves on the plots for the timber column tests (Figs. 140 and 141) carry the designation *T* were located 13 inches below the cap bearing, the lateral positions being evident from the location diagrams, as are also the locations of the couples on the metal of the caps and pintles. The other vertical positions were the same as in the other tests.

In tests Nos. 10 and 11 (Fig. 92) and No. 13 (Fig. 93) the prime mark (′) on one couple designation in each test indicates that the given couple had no asbestos and sheet-iron protection over it.

In test No. 21 (Fig. 96) the prime marks on two curves indicate couples whose leads for 19 inches immediately above the junction were protected by strips of asbestos board 1 inch wide and ¼ inch thick, the object being to determine what influence possible heat interchange between the leads and the junction would have on the indication of the couple.

(*c*) CONNECTIONS TO INSTRUMENTS

The wires of the column thermocouples led from the outlet in the column head above the furnace roof to the junction box placed to one side of the latter (Fig. 31), where they were put under binding posts on a set of connecting blocks. Duplex leads of insulated iron and constantan wire were permanently

installed from the junction box to the bus-bar board in the case housing the recording potentiometer, from whence copper leads extended to the dial switch. The column couples could accordingly be connected with either the indicator or the recorder.

4. METHOD OF TAKING AND REDUCING OBSERVATIONS

In most tests the rate of temperature change was not more rapid, but that readings of both furnace and column thermocouples could be taken on the indicating potentiometer. Since it was found that the observations taken with it were more readily reduced to final form than the recorder records, the temperature determinations as given in the tables and plots are in general based on indicator readings, although the recorder was connected with the furnace couples in practically all tests, and its records served as a general check on temperatures during the test, and were also used in making occasional interpolations between indicator readings in plotting the results.

In taking observations with the indicating potentiometer the electromotive forces of the furnace or the column couples were read in succession in a chosen order, then repeated in reverse order, with equal time intervals (10 seconds or less) between consecutive readings. Mean values were taken of the direct and reverse readings on the respective couples, which, on the assumption of linear temperature change, are equivalent to simultaneous readings on all couples. The readings of the copper-constantan couples at the cold junctions of the furnace pyrometers, were reduced to the corresponding platinum-platinum rhodium electromotive forces, and added to the indications of their respective furnace couples. In the case of the iron-constantan column couples the cold junction correction, which was due to temperature differences between the junctions at the bus-bar board and the zero of the calibration plot, were set off directly on the compensator. For the nickel-constantan column couples a further correction was applied due to the electromotive force introduced by the couple formed in connecting the nickel couple wire to the iron wire leading from the junction box above the furnace to the instruments.

After the proper corrections to the observed electromotive forces had been applied the corresponding temperatures were obtained from the calibration plots.

5. CALIBRATION OF THERMOCOUPLES

(*a*) FURNACE THERMOCOUPLES

The furnace couples, in addition to being compared several times during the series of tests with a standard couple, were frequently submitted to a homogeneity test. The latter is, in effect, a determination of the electromotive force, at frequent points along the couple wire, against standard wire of the same kind. This test was necessary, since the electromotive force of a nonhomogeneous couple—that is, one whose thermoelectric properties vary from point to point along the wire—depends not only on the temperature of the junction, but also on the magnitude and position of the temperature gradients along the length of the couple, which, in general, were not the same as placed in the column-testing furnace as in the furnace used for calibrations.

Changes in calibration were found to be very small as long as the protecting tubes served their intended purpose of excluding the furnace gases. When a protecting tube was broken, or the couple wires had in any manner been exposed to the furnace gases, a homogeneity test was made. When such test showed no electromotive force greater than the equivalent of 15° C at 1000° C, the couple was used again without further treatment. When this value was exceeded, the wire was either brought sufficiently near to its original (uncontaminated) condition by annealing for several hours at 1500° C by passing an electric current through it, or the bad portion of the wire was removed and good wire substituted.

An accuracy within 1 per cent of the indicated temperatures was attained in the great majority of tests. In the case of individual couples in a few tests the homogeneity determination indicated possibility of errors as high as 5 per cent, although the actual errors incurred were probably considerably smaller, since the nonhomogeneous portion of the couple was not necessarily in the region of maximum temperature gradient.

(*b*) COLUMN THERMOCOUPLES

Practically all of the wire used in constructing the column couples was bought in two lots, one of which was purchased in 1914, the other in 1917.

A preliminary study of the variations in the thermoelectric properties among a number of samples taken from the wire in

the first lot indicated that to secure the desired accuracy calibration of individual couples made from this wire would be necessary and that differences between the indications of individual couples were approximately proportional to the temperature.

The latter result showed that a good estimate could be made of the calibration of a given couple by comparing it at one temperature with a standard, and this practice was, in general, followed for all the couples made from the wire in the lot first purchased. Tests on about 50 representative couples made from the wire in the second lot showed sufficiently small variations among the individual couples as to permit the use of one temperature-emf curve for all the couples made from this lot.

The results of the large number of tests made in studying the wire used in constructing column couples indicate that the accuracy was within 2 per cent of the indicated temperatures.

VIII. DEFORMATION MEASUREMENTS

1. GENERAL OUTLINE

Included under this head are means for measuring (1) the unit compression and expansion over a definite gage length, (2) the total depression or expansion of the column measured at a point above its heated portion, (3) the lateral center deflection.

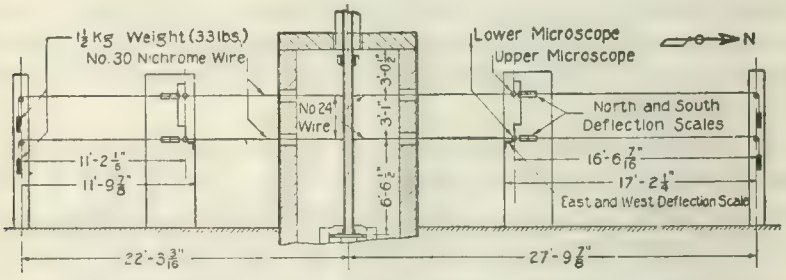


FIG. 33.—*Diagram showing method used for measuring deformation*

The general method used for measuring unit deformation and center deflection is shown in diagram in Fig. 33 and a view of the furnace with attachments in place is given in Fig. 34. Fine nickel-chromium alloy wires were connected to opposite sides of the test column at points symmetrical with its center, using for each a short length of heavier wire and a threaded insert. The

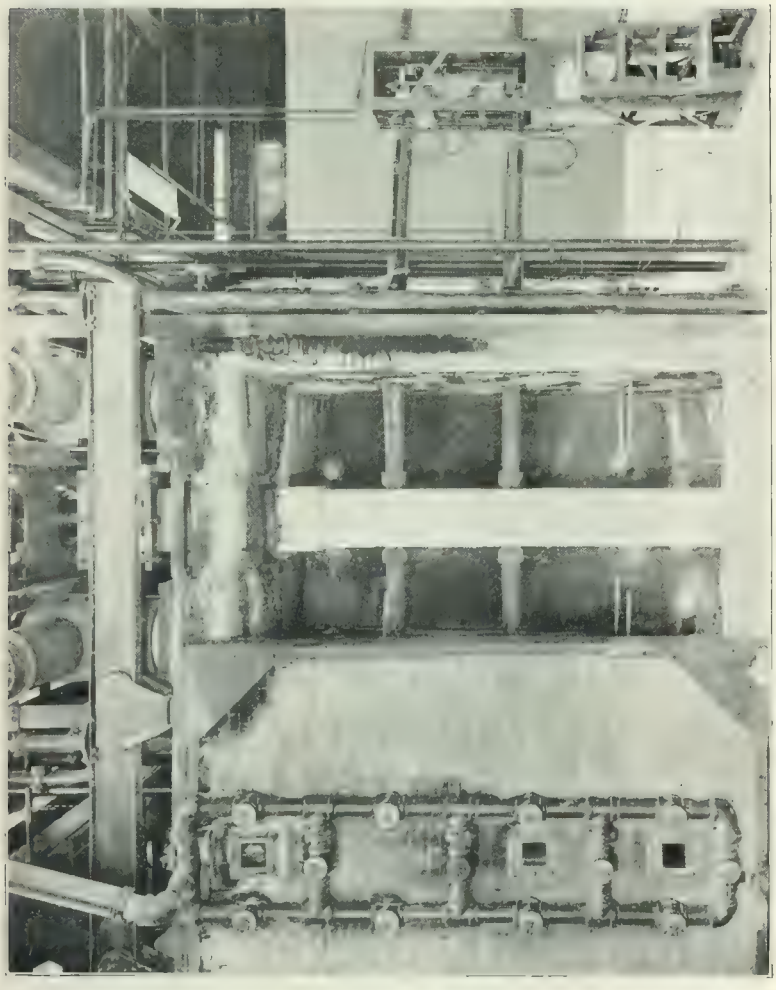


FIG. 34.—*Testing furnace with furnace couples, column, and deformeter in place*

other ends of the wires were weighted and passed over small flanged wheels located in the end posts. The movement of the wires was measured at an intermediate station on each side of the furnace.

(*a*) ATTACHMENT AND PROTECTION OF WIRES

A detail of the insert used for attaching the wire to the column is given in Fig. 35, as are also details of the water-cooled insulating tube. The hole for the insert was prepared before the column was covered or placed in the furnace and a 1¼-inch pipe with collar was bolted around it to provide a clear space for the wire. The pipe projected outside of the covering to afford support for the insulating tube. With the column in place in the furnace, the insert, with wire attached and held in a suitably formed tool, was threaded into the column flange and turned so that the wire was left supported on the upper ledge of the insert.

The insulating tube was designed to protect the wires from the heat of the furnace, as otherwise they would break under the tension to which they were subjected. The circulating water was contained in the annular space between the 2 and 3 inch wrought pipes, the outer pipe being further protected by fire-clay tubing. A 5-inch wrought pipe, with the annular space between it and the 3-inch pipe filled with finely crushed fire brick, was substituted for the fire-clay tubing in the later tests of the series.

Outside of the furnace the wires were shielded by sheet-metal tubes.

2. UNIT COMPRESSION AND EXPANSION

(*a*) MICROMETERS AND MOUNTING

The vertical movement of the wires was measured by means of microscopes set in micrometer slides (Fig. 36). The latter were mounted one on each end of nickel-steel bars supported near the middle. The supports were provided with pivots and slides to secure angular, vertical, and horizontal adjustment of the bar as a whole. The micrometer heads are graduated to 0.005 mm and can be read to the nearest 0.001 mm, the total range of the slide being 50 mm.

(*b*) METHOD OF TAKING AND REDUCING OBSERVATIONS

Readings were taken at the same time on both sides of the furnace, the lower micrometer, upper micrometer, and again, the

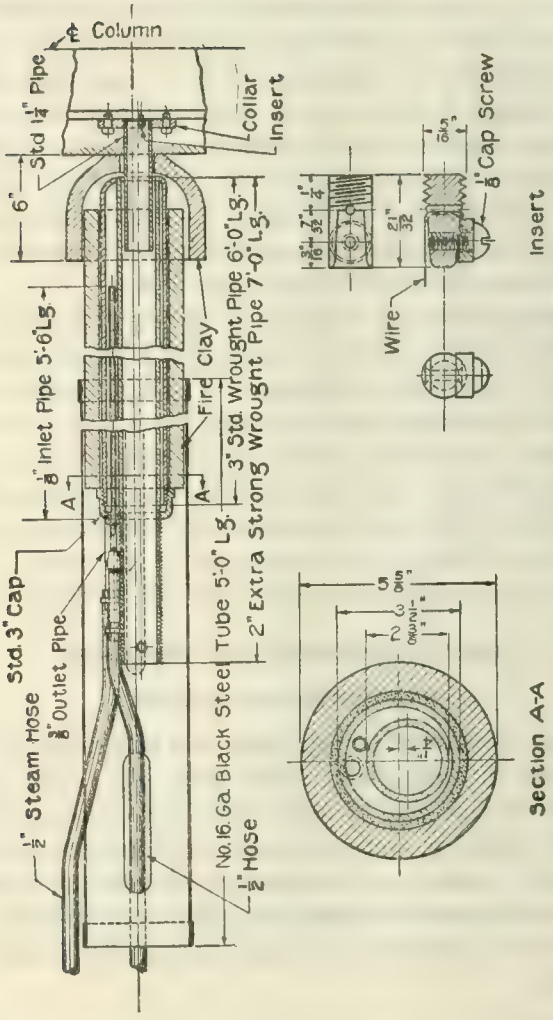


FIG. 35.—*Detail of insulating tube and insert*

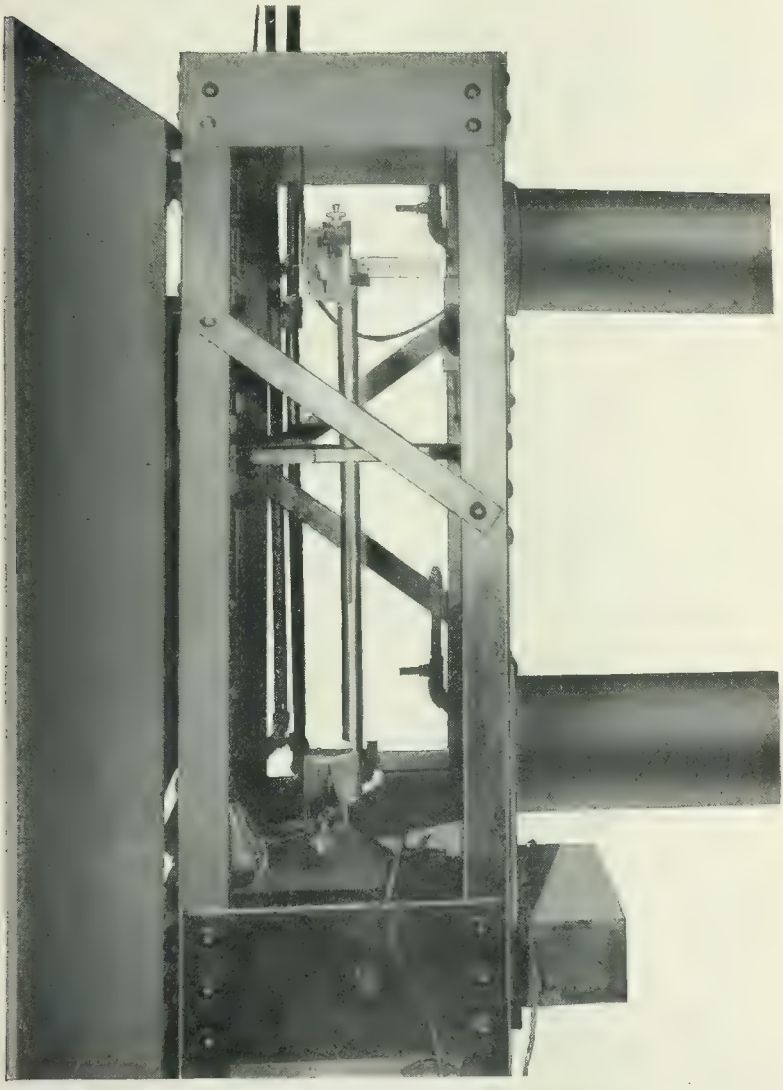


FIG. 30.—*Apparatus for measuring deformation*

lower micrometer being read in the given order at equal intervals. The compression or expansion in the gage length was obtained by multiplying the movement of the upper wire relative to the lower wire at the point of observation by the ratio the distance from the column to the wire support in the end post has to the distance from the latter point to the microscope (Fig. 33). The unit deformation is the average of the values of total deformation obtained as given above, divided by the distance between gage points (94 cm).

3. CENTER DEFLECTION

The lower gage point was located near the center of the heated length of the column, and the center lateral deflection was measured by means of polished nickel scales, readings being taken with reference to points on the wires and their reflection in the scales.

The east-and-west deflection scales were placed perpendicular to the wires, and the deflection of the column was obtained by multiplying the observed movement at the scale by the ratio of distances, as given in Fig. 33, of the column and of the scale, respectively, from the wire supports in the end posts, taking the average of values obtained on the two sides.

The north-and-south deflection scales were placed parallel with the wires, small flat disks being attached to the latter in front of the scales, with reference to which readings were taken. The deflection in the north or south direction is the average of the movements observed at the two posts, assuming the expansion of the measuring wires on both sides of the column to be equal.

4. TOTAL DEPRESSION OR EXPANSION

The total vertical depression or expansion of the column before failure, as indicated by the movement of the head of the loading ram, was determined for the columns in the fire and water test series, the timber columns, and a few subsequent tests of structural-steel and cast-iron columns. For this purpose a single No. 30 wire on one side of the furnace only was attached to the bearing below the head of the ram plunger, the details of attachment and measurement of its movement being the same as described in paragraph 2 of this section.

5. CALIBRATION AND ACCURACY

Calibrations of the instruments and appliances used for measuring unit deformation and study of the test results, indicate possible maximum errors of 20 parts in 100 000 (0.0002), which correspond to errors in instrument readings of about 0.008 inch (0.2 mm). These errors were caused mainly by the change in sag of the wires due to moisture in the insulating tubes and also by movement of the column as a whole in the time interval required by a set of micrometer readings. The errors due to irregularities in micrometer screws and slides and in parallelism of slides and of microscopes as mounted on the supporting bar were relatively small.

In determining the total expansion or depression of the column by measurement of the movement of the loading ram a large error was incurred in applying the load due to deflection of supports. During the subsequent fire exposure under constant load the supports appear to have remained fairly rigid, the maximum errors for this period being probably within 0.01 inch (0.25 mm) for the steel and cast-iron columns and 0.04 inch (1 mm) for the timber columns.

The center deflection in the east-and-west direction was determined with an accuracy of about 0.04 inch (1 mm) and in the north-and-south direction within about 0.08 inch (2 mm). The principal source of error in the latter case was unequal expansion of the wires due to unequal temperatures within the insulating tubes.

IX. METHOD OF TESTING

The columns in the fire-test series were subjected to a constant working load and fire exposure increasing according to a predetermined time-temperature relation until failure occurred or until they had withstood the test eight hours, or more.

In the fire and water tests the working load was maintained constant and the column exposed to fire for a predetermined period when water at given pressures was applied by means of a hose stream.

1. LOADING FORMULAS AND APPLIED LOADS

(a) WORKING LOADS

The formulas used in calculating the working loads to be placed on the columns were among those in most general use at the time the method of procedure for these tests was determined. They are given in Table 41, where are also given the properties of the columns and the calculated and applied working loads.

The applied loads, which include weights of the column head and bearing blocks (2500 pounds), exceed the calculated working loads

TABLE 41.—COMPUTED AND APPLIED WORKING LOADS

SECTION	Formula	Nominal Area, Sq. In.	l/r	Computed Load		Applied Load	
				Lb. per Sq. In.	Total, Lb.	Total, Lb.	Per cent of Computed Load
Rolled H	16000–70 l/r	10.17	75.6	10710	108900	119500	109.7
Plate and Angle	do	13.00	111.8	8170	106200	116000	109.2
Plate and Channel	do	8.76	64.7	11470	100500	111000	110.5
Latticed Channel	do	7.78	44.0	12920	100500	111000	110.5
Z-bar and Plate	do	9.32	81.7	10280	95900	105000	109.5
I-beam and Channel	do	10.12	72.1	10950	110800	122000	110.1
Latticed Angle	do	8.44	40.7	13160	111000	122500	110.4
Starred Angle	do	13.27	108.5	8435	111900	124000	110.7
Round Cast Iron (Vertically Cast)	10000–60 l/r	14.45	63.2	6210	89700	98500	109.8
Round Cast Iron (Horizontally Cast)	do	14.73	68.2	5910	87100	95500	109.6
Round Cast Iron (Concrete filled)	do	14.73	68.2	5910	87100	95500	109.6
			l/d				
Steel Pipe (Concrete filled)	As (13500–140 l/d) +Ac (1000–11 l/d).	Steel 6.93 Concrete 38.74	19.6	Steel 10750 Concrete 784	104900	114500	109.2
Reinforced Steel Pipe (Starred angles embedded in the concrete filling)	do	Steel 18.36 Concrete 40.07	17.7	Steel 11060 Concrete 805	234700	236000	100.6
Square Vertically Reinforced Concrete	450 (Ac+15 As)	Concrete 140 Steel 4.00	12.7	Concrete 450 Steel 6750	90000	101000	112.2
Round Vertically Reinforced Concrete	450 (Ac+15 As)	Concrete 127 Steel 6.00	11.7	Concrete 450 Steel 6750	97500	107500	110.3
Hooped Reinforced Concrete	650 (Ac+15 As)	Concrete 129 Steel 3.38	11.7	Concrete 650 Steel 9750	117000	129000	110.3
Timber	$1000\left(1-\frac{l}{80d}\right)$	129.4	13.4	833	107700	118500	110.0

l—Effective length, inches.
r—Least radius of gyration, inches.
d—Diameter or side, inches.
As—Area of vertical steel, sq. inches.
Ac—Area of concrete, sq. inches.

by about 10 per cent for most of the columns. The principal part of the excess was incurred in the first tests of the series which were made before the calibration of the loading ram was completed. It was decided to maintain the same loads for the subsequent tests as representative of a moderate condition of overload.

No allowance was made for the load-carrying capacity of the covering or filling in any of the tests except in case of the pipe columns where the loads were calculated according to the loading formula in use by the manufacturer who supplied the columns.

(*b*) LOADING TO FAILURE

In case the column withstood the 8-hour fire test it was immediately loaded to failure under full fire exposure.

In the fire and water test series, three protected structural steel, the two unprotected cast-iron, and the three reinforced-concrete columns were loaded to failure after they had cooled. Four protected structural-steel columns after subjection to fire and water application, were loaded to a little over twice the applied working load and reserved for use in further tests.

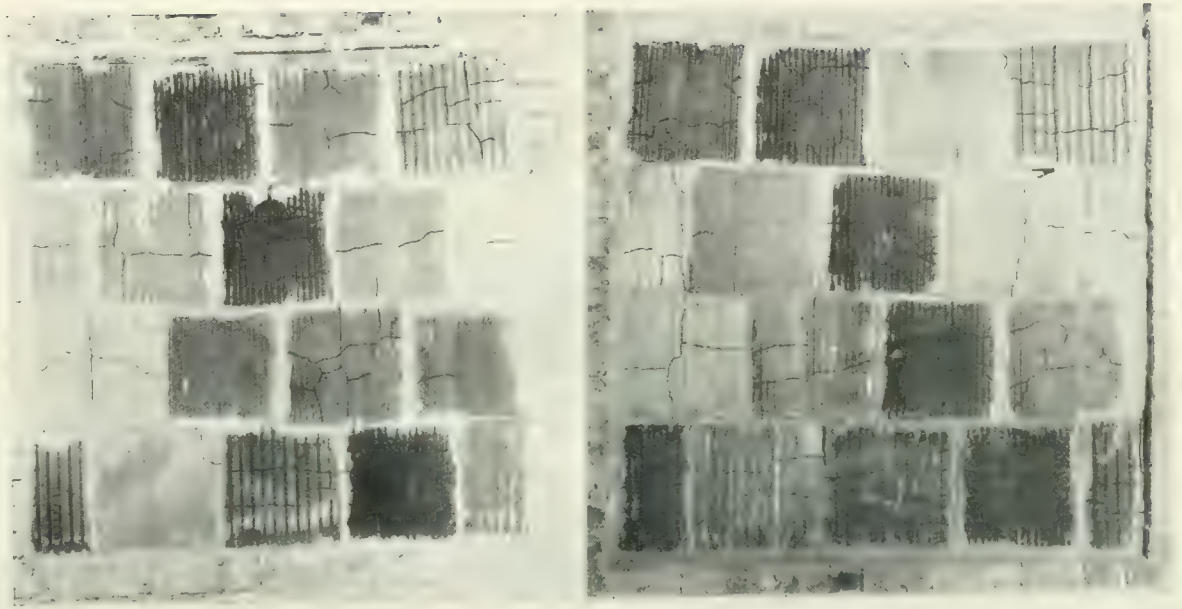
2. FIRE EXPOSURE

(*a*) CHARACTER OF THE FIRE

The fuel used was carbureted water gas from the city service mains. It was admitted at the primary burners with insufficient air for immediate complete combustion, in order to avoid excessive temperatures at the bottom of the furnace and allow further combustion to take place at the secondary air inlets. The gas was generally burned with sufficient air to prevent deposit of soot on the test column or furnace walls.

(*b*) PRELIMINARY PANEL TESTS

During the first half hour it is not in general possible to regulate the temperature of a large testing furnace to correspond closely with predetermined temperatures; also, the determination of temperature is complicated by the lag of the furnace pyrometers with the rapid temperature rise occurring during this period. To obtain information on the effect of varying the rate of temperature rise, two panels built up of the five kinds of hollow clay tile used as protection in the column tests, were subjected to fire on one side for one-half hour and allowed to cool in place. The



Panel No. 1 Panel No. 2

Black lines trace fine cracks developed in the test

FIG. 38.—*Preliminary tile panels after test*

average furnace temperatures obtaining in the two tests are given in Fig. 37, and in Fig. 38 the condition of the panels after test is shown. Most of the tile developed fine cracks which extended to the inside of the inner shell and in a few cases through the cross webs to the outer shell. Cracks along the cross webs near the inner shells were common and some inner shells were loose. While in one test, the indicated furnace temperature rise in the first 10 minutes was over twice as rapid as in the other, the condition of the two panels after test was nearly the same.

(c) TIME-TEMPERATURE CURVE

The temperature of the furnace was regulated to conform as nearly as practicable with a predetermined time-temperature reference curve shown in broken lines in Fig. 39. Here is also shown the furnace temperature attained, the curve being obtained by

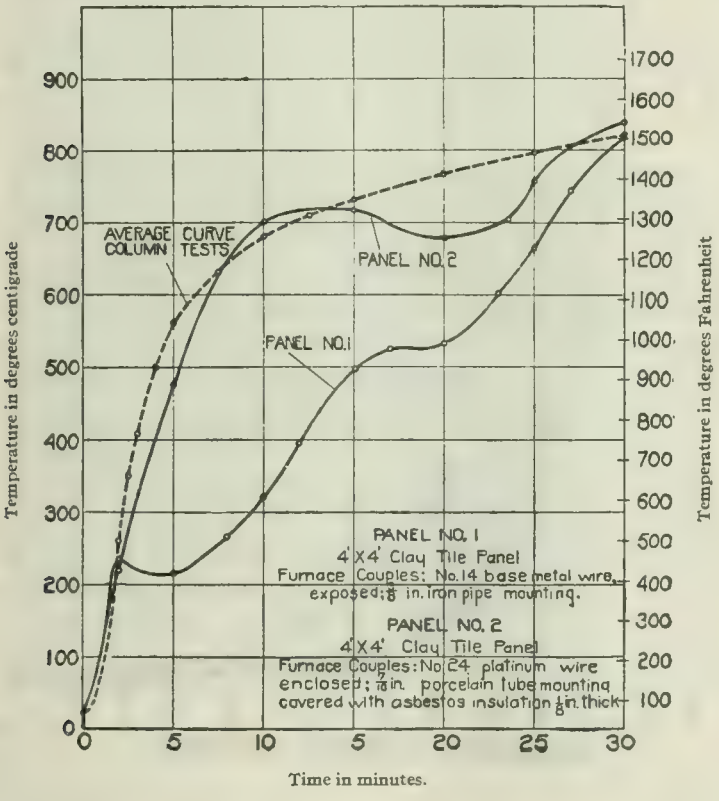


FIG. 37.—*Furnace temperatures, preliminary panel tests*

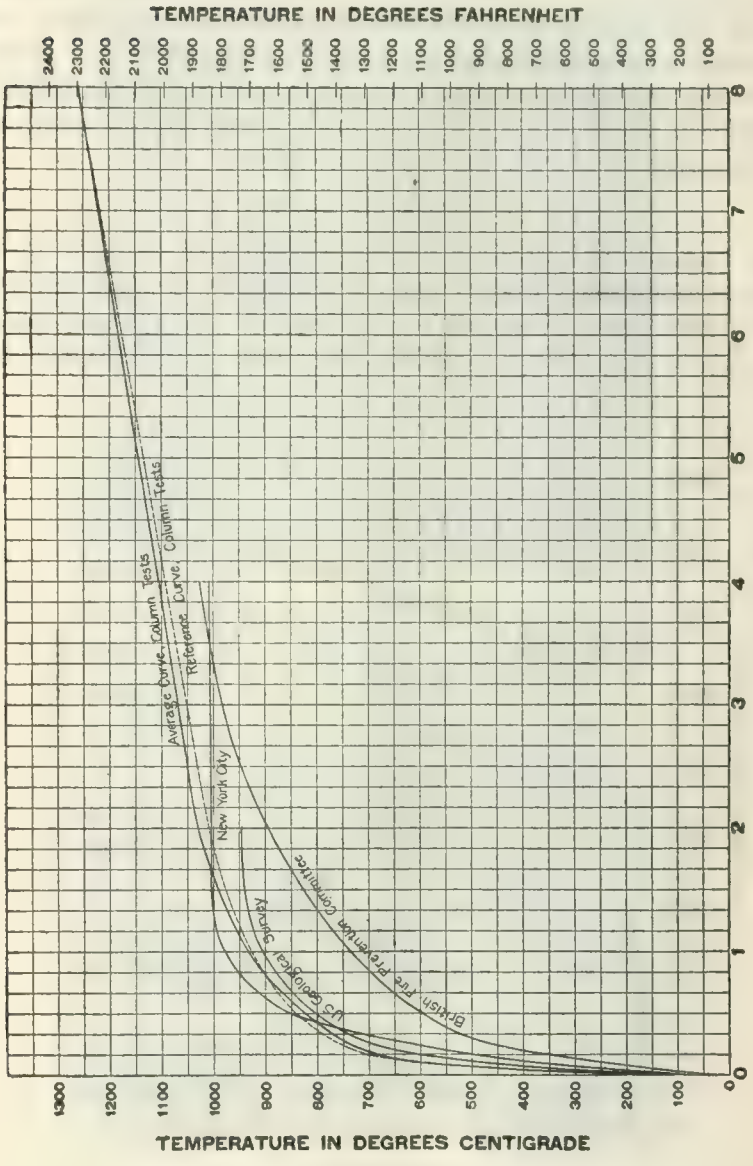


FIG. 39.—*Comparison of furnace temperatures used in fire tests*

averaging the results from the full number of tests. Three curves giving the average indicated furnace temperatures attained in previous fire tests are also given.[2]

On the reference curve of the column tests a furnace temperature of 832° C (1550° F) obtains at the end of 30 minutes, and 927° C (1700° F) at the end of the first hour. From this point the temperature increases at a gradually varying rate up to 2 hours, when the temperature is 1010° C (1850° F). Subsequent to this the rate of rise is constant, 1101° C (2000° F) being attained at the end of 4 hours, and 1260° C (2300° F) at the end of 8 hours. The average indicated furnace temperature was somewhat below the reference curve in the first hour and above it by a maximum of 25° C during the subsequent periods.

(*d*) INFLUENCE OF PYROMETER MOUNTING ON INDICATED TEMPERATURES

With the type of mounting it was deemed necessary to use in the column tests (Fig. 32) the pyrometers did not indicate the full measure of the temperature effects within the furnace chamber, due mainly to the heat insulating and radiating properties of the mounting.

(1) TEMPERATURE LAG.—The retardation due to the heat insulation given the thermocouple by the mounting was particularly marked during the first part of the test period, when, with the obtaining rapid temperature rise, the indicated temperatures were considerably lower than those acting on the mounting on account of the time required to produce temperature equilibrium throughout the latter. This effect will be termed "lag," and a measure of its extent can be obtained by application of the principles governing heat interchange.

When the rate of temperature rise is not too rapid and the heat interchange is chiefly due to conduction, within a given range, the temperature lag is very nearly proportional to the rate of rise of indicated temperature, or,

$$U - \theta = \lambda \frac{d\theta}{dt} \qquad (1)$$

where θ is the temperature indicated by the pyrometer, U is the

[2] cf. Woolson and Miller, Paper No. 27_2, International Association for Testing Materials, 1915; floor test Nos. 55 to 74. Humphrey, U. S. G. S. Bulletin No. 370; panel test Nos. 1 to 30. British Fire Prevention Committee, Journal No. 6, 1911; average from eight full-protection, Class B, floor tests.

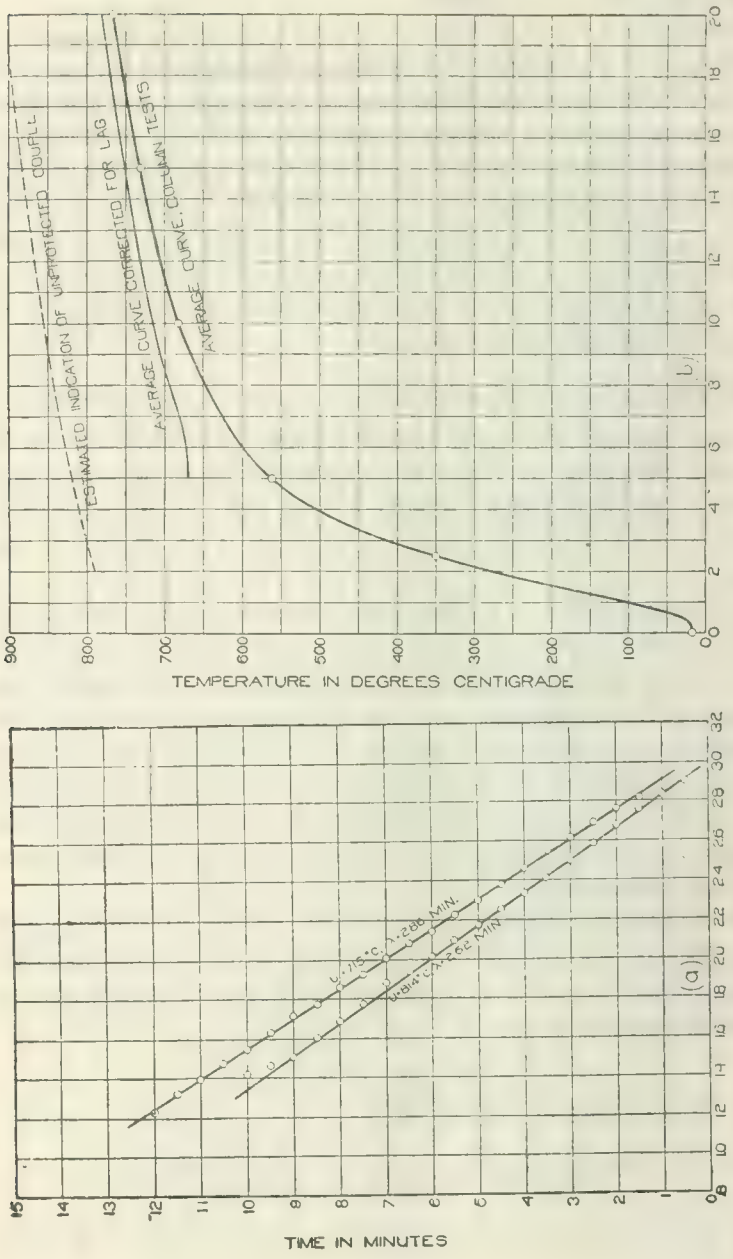


FIG. 40.—*Lag correction for furnace pyrometers*

temperature the pyrometer would indicate if the furnace conditions at the given instant were maintained constant a sufficiently long time, $\frac{d\theta}{dt}$ is the rate of rise of indicated temperature, and λ is a constant for the pyrometer considered within a given range in U. To determine λ, the pyrometer was plunged into a gas-fired furnace whose temperature was maintained as nearly constant as possible and readings taken until the indicated temperatures showed little or no change. The experiment was begun as soon after the gas was lighted as was consistent with maintaining steady temperature in order that the furnace conditions might approximate those present during the first part of a column test. If U, for the conditions of the experiment, be assumed constant, from equation (1)

$$\lambda = \frac{t}{\log_e (U-\theta)} + C, \qquad (2)$$

where t is the time after the pyrometer was introduced into the furnace and C is the constant of integration, which latter, in the method applied, is eliminated by subtraction.

The results of two determinations are given in Fig. 40 (*a*), λ being obtained from the slope of the line connecting the experimental points, multiplied by 0.4343, the modulus of the common system of logarithms. As applied for correcting the average curve of indicated temperature of the column tests, a lower value than the average experimental result was used to allow for impaired and thinner insulation that was present in a number of tests, a value of λ of 2.4 minutes being used up to indicated temperatures of about 800° C. Determinations made at higher temperatures gave lower results, values of about 2 minutes obtaining for mountings with unimpaired insulation tested at 950 to 1000° C.

In Fig. 40 (*b*) the lower curve is the average curve of indicated furnace temperature of the column tests for the first 20-minute period. Corrections for lag were deduced for the indicated temperatures given by the curve in the interval 5 to 20 minutes by substituting the given value of λ in equation (1), the rate of rise, $\frac{d\theta}{dt}$, being obtained from tangents drawn to the average curve at points ½ to 1 minute apart. As given by the difference between the original and corrected curves, the lag decreases from a little over 100° C at 5 minutes to 14° C at 20 minutes. From there on

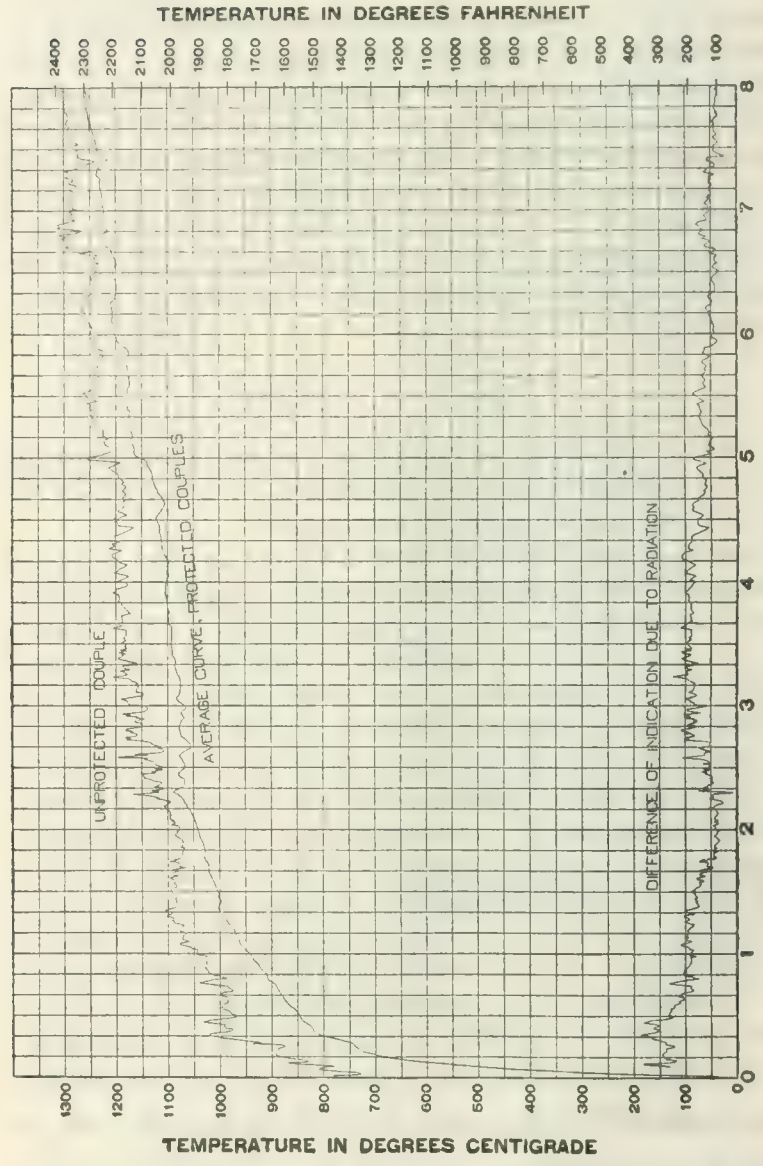


FIG. 41.—*Effect of radiation on indication of furnace pyrometers*

the decrease is gradual up to the uniform slope after two hours (Fig. 39), where the lag correction is about 1° C. For the furnace temperatures of the individual tests the corrections would be much more irregular than those on the average curve on account of irregularities of slope, although, in general, after the first hour they would be relatively small and within the limit of error applicable to furnace-temperature measurements. The furnace temperatures as given on the plots in Figs. 90 to 145 are indicated temperatures, as it was not deemed practicable to apply corrections for either lag or radiation effects to results of individual tests.

(2) Radiation Effects.—In a gas-fired furnace where the source of heat is combustion within the chamber the inside of the walls of the latter will be at lower temperature than the furnace contents. The indications of a pyrometer introduced into the chamber to measure the temperature of its contents will be influenced by radiant interchange of heat with the inclosure, the extent of the effect being dependent mainly upon the size of the pyrometer and the temperature difference between the inclosure and the gaseous contents. To obtain a measure of the extent to which the indications of the furnace pyrometers were thus affected, an unprotected thermocouple of fine platinum—platinum-rhodium wires 0.004 inch (1/10 mm) in diameter was supported 1½ inches outside of the closed end of a porcelain tube on similar wires of larger diameter that were passed through and sealed into the end of the tube, through which they led to the outside of the furnace. Two pyrometers of the same design as those used in the column tests (Fig. 32) were placed about 3 inches away from the unprotected couple and several runs made in a furnace having walls of the same material and of about the same thickness as those of the column furnace. Readings of the unprotected and the protected couples were taken at intervals of one-half minute or less. The indicated temperatures obtained in a run extending to 8 hours are plotted in Fig. 41. Those given by the unprotected couple were considerably higher than those of the protected couples, because the temperature of the fine wire was influenced to a much smaller extent by radiant heat interchange with the inside of the furnace inclosure than that of the protected couples, and the temperatures indicated by it approximate those of the furnace contents as closely as can be attained without excessive refinements.

The difference in indication due to radiation, as given by the lower curve in Fig. 41, was obtained by correcting the average indication of the protected couples for lag and subtracting the corrected value from that of the unprotected couple. While the variations in the resulting difference of indication are partly due to local and general variations in furnace conditions and minor changes in calibration of the unprotected couple, the general decrease from about 150° C during the first half hour to less than 50° C at the end of 8 hours can be ascribed mainly to decrease in temperature difference between the furnace contents and the inside surface of the furnace chamber.

The average difference due to radiation was applied to the results plotted in Fig. 40 (*b*), and the estimated indication of the unprotected fine-wire couple is given by the upper curve.

Since the indication of all pyrometers is influenced by lag and radiation effects, the comparison of furnace exposures afforded by the indicated temperatures given in Fig. 39 is limited by differences in these particulars incident with the types of pyrometers used.

3. FIRE AND WATER TEST PROCEDURE

The duration of the fire periods varied from 22½ minutes to one hour, and that of the subsequent water application from one to five minutes. The length of the maximum fire period was determined by the time within which water is generally applied in building fires, which was estimated as being one hour. The unprotected columns and some of the protected columns were given fire periods of shorter duration which were well within the time to failure of the corresponding columns in the fire-test series. The duration and pressure of the water application were also varied for the different types of protection, the heavier ones being subjected to the most severe test conditions.

In applying the hose stream, the nozzle was moved back and forth on one side of the furnace and maintained at a constant distance from the column, the water being applied in succession over the full height on three of its sides.

4. OBSERVATIONS DURING TEST

Readings for temperature of furnace and of test column were taken at intervals of 2 to 15 minutes, the frequency of the readings

depending on the rate of temperature change. Readings for deformation were also taken at intervals of 2 to 15 minutes.

Notes were taken on the character of the fire and on its effects on the test column at all stages of the test where the latter could be observed.

5. OBSERVATIONS AFTER FAILURE

After the column had cooled notes were taken of its general condition and the covering was partly or wholly removed to determine the extent to which it was damaged. Measurements were taken of the amount and direction of the final buckle.

6. PHOTOGRAPHIC RECORDS

Photographs were taken of all columns after test and also of a number of typical columns before test. The views were generally taken diagonally and from opposite sides, so as to show all faces of the column, only one view being as a rule included in this paper.

X. RESULTS OF FIRE TESTS

1. FIRE-TEST RESULTS IN TABLES AND FIGURES

The results of the fire tests in points of time to failure, load sustained, and relative fire exposure are given in Tables 42*a* to 42*i*. The columns are grouped by classes of protection, and the thickness and material applied in each test are indicated.

In Table 43 (pp. 122–123) are given the period of expansion, the time to failure, and the maximum temperatures attained in the metal.

The period of expansion of a column when loaded and exposed to fire is the period from the beginning of the test to the time when expansion of the column ceases, due to yielding of the heated metal under the applied load.

The time to failure in the fire test extends from the beginning of the test to the time when the column is unable to sustain the applied working load.

The time to failure of all columns in the fire-test series is given in diagram form in Fig. 42 (p. 114), and the period of expansion of the steel, cast-iron, and concrete columns is given in similar manner in Fig. 45 (p.121).

2. PHOTOGRAPHIC RECORDS

Views before and after test of the columns in the fire-test series are given in Figs. 58 to 82, Appendix A (pp. 216–241).

TABLE 42a.—RESULTS OF FIRE TESTS

Unprotected columns

Test No.	SECTION	Nominal Area, Sq. In.	l/r	Load Sustained During Test		Time to Failure, Hr—Min.	Furnace Exposure, Percent
				Total Load, Lb.	Unit Load, Lb. per Sq. In.		
1	Rolled H	10.17	75.6	119500	11750	0—11¼	95.6
2	Plate and Angle	13.00	111.8	116000	8900	0—19¼	97.0
3	Plate and Channel	8.76	64.7	111000	12650	0—14	101.4
4	Latticed Channel	7.78	44.0	111000	14250	0—11	101.1
5	Z-bar and Plate	9.32	81.7	105000	11250	0—14¼	100.0
6	I-beam and Channel	10.12	72.1	122000	12050	0—17	89.6
7	Latticed Angle	8.44	40.7	122500	14500	0—14	96.3
8	Starred Angle	13.27	108.5	124000	9350	0—21½	99.0
9	Round Cast Iron	14.73	68.2	95500	6500	0—34¼	101.2
10	Round Cast Iron	14.73	68.2	95500	6500	0—34½	101.7
10A	Round Cast Iron	14.45	63.2	98500	6800	0—34¼	100.3
11	Round Cast Iron(Concrete filled)	14.73	68.2	95500	.....	0—45¼	103.1
12	Steel Pipe (Concrete filled)	Steel 6.93 Concrete 38.74	63.9	114500	.....	0—36	101.9
3	Reinforced Steel Pipe (Starred angles imbedded in concrete filling)	Steel 18.36 Concrete 40.07	68.2	236000	.....	1—11¾	99.3

NOTE: Details of the steel, cast-iron, and pipe columns listed in Tables 42*a* to 42*g* are given in Figs. 1 to 4 (p. 16-19), and Figs. 6 and 7 (p. 22 and 24). Further properties of the sections are given in Tables 3*a* to 3*h* (p. 33-47).

TABLE 42b.—RESULTS OF FIRE TESTS

Columns partly protected by concrete

Test No.	SECTION	*Protection		Age of Covering, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
		Mixture	Kind of Concrete				
14	Rolled H.....	1:2:4	Joliet gravel.....	405	119500	1 — 04¼	85.1
15	Rolled H.....	1:2:4	Rockport granite....	407	119500	0 — 48¼	91.5
16	Plate and Angle...	1:2:4	New York trap.....	416	116000	0 — 44½	94.3
17	Plate and Angle...	1:1½:4½	Hard coal cinders...	408	116000	0 — 41¾	99.6
18	Latticed Channel..	1:2:4	New York trap.....	418	111000	2 — 53	98.1
19	Z-bar and Plate....	1:3:5	Chicago limestone...	414	105000	1 — 07¼	95.8
20	I-beam and Channel.....	1:3:5	New York trap.....	415	122000	1 — 24½	100.2
21	I-beam and Channel.....	1:3:5	New York trap.....	416	122000	1 — 21¾	99.6
22	Latticed Angle....	1:2:4	Chicago limestone...	408	119500	5 — 14	99.2

*Re-entrant portions and interior filled with concrete.

TABLE 42c.—RESULTS OF FIRE TESTS

Columns protected by plaster on metal lath

Test No.	SECTION	Protection	Age of Covering, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
23	Plate and Angle...	Two 2-coat layers of Portland cement plaster on expanded metal lath, each layer 1 in. thick, with a ¾-in. air space between layers	508	*117500	2 — 52	103.1
24	Plate and Channel.	Two 2-coat layers of Portland cement plaster on woven wire lath, each layer ⅞ in. thick with a ¾-in. air space between layers	496	111000	2 — 24	101.5
25	Z-bar and Plate...	One 2-coat layer of Portland cement plaster, 1 in. thick, on expanded metal lath	484	105000	1 — 07¾	103.7
26	Latticed Angle....	One 2-coat layer of Portland cement plaster, 1⅛ in. thick, on expanded metal lath	497	122500	1 — 23½	104.2
27	Round Cast Iron..	One 2-coat layer of Portland cement plaster, 1½ in. thick on high ribbed expanded metal lath with a ½-in. broken air space	498	95500	2 — 58	98.2

* Heavier load used as plate has 1/32 in. greater thickness than nominal.

TABLE 42d.—RESULTS OF FIRE TESTS

Columns protected by concrete

Test No.	SECTION	Protection			Age of Covering, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
		Thickness, In.	Mixture	Kind of Concrete				
28	Rolled H	2	1:2:4	Chicago limestone	437	119500	6 — 33¾	96.9
28A	Rolled H	2	1:2:4	Chicago limestone	438	119500	7 — 09¼	95.2
29	Rolled H	2	1:2:4	New York trap	435	119500	4 — 38½	99.5
30	Rolled H	2	1:2:4	Joliet gravel	439	119500	7 — 16	99.2
31	Rolled H	2	1:2:4	Cleveland sandstone	500	119500	4 — 11½	99.4
32	Rolled H	2	1:2:5	Hard coal cinders	503	119500	3 — 44	101.3
32A	Z-bar and Plate	2	1:2:5	Hard coal cinders	497	105000	4 — 02	100.5
33	Rolled H	4	1:2:4	Chicago limestone	450	119500 ‡431000	8 — 08	99.2
33A	Rolled H	4	1:2:4	Chicago limestone	455	119500 ‡405000	8 — 07¼	98.8
34	Rolled H	4	1:2:4	Rockport granite	452	119500	7 — 58	98.6
34A	Rolled H	4	1:2:4	Rockport granite	454	119500	7 — 23	102.5
35	Rolled H	4	1:3:5	Chicago limestone	504	119500 ‡348000	8 — 07	101.3
36	Plate and Angle	2	1:2:4	New York trap	445	116000	3 — 53¼	100.3
37	Plate and Angle	4 round	1:2:4	New York trap	503	116000	7 — 34½	99.9
38	Plate and Channel	2	1:2:4	Joliet gravel	449	*116500	5 — 28¾	100.2
39	Plate and Channel	4	1:2:4	Meramec R. gravel	436	*116500	3 — 41¼	98.3
40	Latticed Channel	2 round	1:2:4	New York trap	501	111000	7 — 57	99.1
41	Z-bar and Plate	4	1:3:5	Chicago limestone	451	105000 ‡332000	8 — 24½	99.6
42	Z-bar and Plate	4	1:3:5	Chicago limestone	453	105000 ‡333000	8 — 11½	98.5
43	I-beam and Channel	2	1:2:4	Cleveland sandstone	456	122000	4 — 11	98.7
44	I-beam and Channel	2	1:3:5	Cleveland sandstone	458	122000	3 — 04¼	100.6
45	Starred Angle	2 round	1:2:4	Meramec R. gravel	447	124000	1 — 47	99.7
46	Latticed Angle	†2	1:2:4	New York trap	451	122500	6 — 43½	99.0
47	Round Cast Iron	2	1:2:5	Hard coal cinders	446	95500	2 — 48¾	99.6

* Heavier load used as plates have 1/32 in. greater thickness than nominal.
† 2-in. outside rivets, 3½-in. outside angles.
‡ Load necessary to cause failure of column. After 8 hr. the load was increased until failure occurred.

TABLE 42e.—RESULTS OF FIRE TESTS

Columns protected by hollow clay tile

Test No.	SECTION	Protection		Age of Covering, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure Percent
		Thickness of Tile, In.	Kind of Tile, Filling, and Method of Tying				
48	Rolled H	2	New Jersey semi-fire clay No filling Outside wire ties	496	119500	1 — 50	100.9
49	Rolled H	4	Same as No. 48	497	119500	1 — 40	99.9
50	Plate and Angle	2	Surface clay, Boston Granite concrete fill No ties	494	116000	1 — 05¾	99.8
50A	Plate and Angle	2	Same as No. 50	505	*117500	1 — 59½	101.8
51	Plate and Angle	4	Surface clay, Boston Granite concrete fill Outside wire ties	487	116000	2 — 17¼	101.8
51A	Plate and Angle	4	Same as No. 51	507	116000	2 — 55½	100.5
52	Plate and Channel	2	Ohio shale Cinder concrete fill Outside wire ties	513	111000	1 — 40¾	100.2
53	Plate and Channel	4	Same as No. 52	495	111000	1 — 22¼	103.0
54	Latticed Channel	2	Ohio semi-fire clay Trap concrete fill Outside wire ties	489	111000	3 — 17¼	101.1
55	Z-bar and Plate	2	Ohio semi-fire clay Limestone concrete fill Outside wire ties	485	105000	3 — 46¾	99.5
56	Z-bar and Plate	4	Ohio semi-fire clay Limestone concrete fill Wire mesh in joints	491	105000	3 — 33½	100.3
57	I-beam and Channel	4	Surface clay, Chicago Limestone concrete fill Outside wire ties	491	122000	3 — 23	100.0
58	I-beam and Channel	Two 2	Surface clay, Chicago Hollow tile fill Wire mesh in joints	502	122000	4 — 35¾	101.7
59	I-beam and Channel	Two 2	Surface clay, Chicago Hollow tile fill Outside wire ties	490	122000	1 — 33¾	101.2
60	Latticed Angle	2	Ohio semi-fire clay Trap concrete fill, placed before tile was set Outside wire ties	487	122500	3 — 09½	100.4
61	Latticed Angle	2	Ohio semi-fire clay No filling Outside wire ties	501	122500	0 — 50¼	100.3
62	Round Cast Iron	2 round	Porous semi-fire clay, New Jersey No filling Outside wire ties	483	95500	4 — 11½	101.3
63	Round Cast Iron	2 round	Same as No. 62	493	95500	2 — 57½	100.6
76	Rolled H	2	Ohio shale; Ohio semi-fire clay; semi-fire clay, New Jersey Limestone concrete fill Wire mesh in joints Tile covered with ¾-in. layer of gypsum plaster	42	119500	4 — 25½	98.1
77	Plate and Angle	4	Semi-fire clay, N. J.; surface clay, Chicago; surface clay, Boston. Limestone concrete fill Wire mesh in joints Tile covered with ⅝-in. layer of lime plaster	45	116000	4 — 42¼	97.3

* Heavier load used as plate has 1/32 in. greater thickness than nominal.

TABLE 42f.—RESULTS OF FIRE TESTS

Columns protected by gypsum block

Test No.	SECTION	Thickness of Block, In.	Protection: Kind of Block, Filling and Method of Tying	Age of Covering, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
64	Rolled H..	4	Western gypsum (solid) Hollow gypsum block fill Wall ties in joints	502	119500	4 — 43¼	104.5
65	Plate and Channel	2	Western gypsum (solid) Solid gypsum block fill Wall ties in joints	505	111000	2 — 21½	104.5
66	Latticed Channel	2	Eastern gypsum (solid) Poured gypsum fill Wire mesh in joints	495	111000	2 — 36	101.0
67	Rolled H..	4	Same as No. 66	492	119500	5 — 31½	101.2
67A	Rolled H..	4	Same as No. 66	491	119500	6 — 24½	99.7

TABLE 42g.—RESULTS OF FIRE TESTS

Columns protected by brick

Test No.	SECTION	Thickness of Brick, In.	Kind of Brick and Filling	Age of Covering, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
68	Rolled H..	2¼	Chicago common brick set on edge and end Brick fill	498	119500	1 — 40¾	104 0
69	Rolled H..	3¾	Chicago common brick laid flat Brick fill	502	119500	7 — 13¼	101 2

TABLE 42h.—RESULTS OF FIRE TESTS

Reinforced concrete columns

Effective length, 12 ft., 8 in.

Test No.	SECTION	Outside Dimensions and Reinforcement	Concrete		Age, Days	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
			Mixture	Kind.				
70	Square Vertically Reinforced	16-in. square..... Four 1-in. sq. bars	1:2:4	Chicago limestone	433	101000 ‡294000	8 — 40¼	101.1
71	Square Vertically Reinforced	Same as No. 70....	1:2:4	New York trap	450	101000	7 — 22¾	99.2
72	Round Vertically Reinforced	17-in. diameter.... Six 1-in. sq. bars	1:2:4	Chicago limestone	520	107500 ‡250000	8 — 04½	102.1
73	Round Vertically Reinforced	Same as No. 72....	1:2:4	New York trap	442	107500	7 — 57½	98.5
74	Hooped Reinforced	17-in. diameter.... Six ¾-in. sq. bars ¼-in ϕ spiral on 1½-in pitch	1:2:4	Chicago limestone	522	129000 ‡243000	8 — 06½	99.5
75	Hooped Reinforced	Same as No. 74....	1:2:4	New York trap	460	129000 ‡163000	8 — 01¾	100.7

‡Load necessary to cause failure of column. After 8 hr. the load was increased until failure occurred.

TABLE 42i.—RESULTS OF FIRE TESTS

Timber columns

Nominal net section, 11⅜ by 11⅜ in. Effective length, 12 ft., 8½ in.

Test No.	Species	Bearing Details	Protection of Column and Cap	Load Sustained During Test, Lb.	Time to Failure, Hr.—Min.	Furnace Exposure, Percent
78	Longleaf pine.	Cast iron cap and pintle	One 2-coat layer of Portland cement plaster 1 in. thick on woven wire lath, ¾-in. air space. Age, 30 days	118500	2 — 15¼	97.6
79	Longleaf pine.	Cast iron cap and pintle	Unprotected.....	118500	0 — 50	105.2
80	Longleaf pine.	Steel plate cap and timber strut	One thickness of ⅜-in. gypsum wall board with metal corner beads nailed to column	118500	1 — 13	103.5
81	Longleaf pine.	Steel plate cap and timber strut	Unprotected.....	118500	0 — 35	113.0
82	Douglas fir....	Cast iron cap and pintle	Unprotected.....	118500	0 — 45¼	106.6
83	Douglas fir....	Steel plate cap and timber strut	Unprotected.....	118500	0 — 38½	106.1

NOTE: Details of design of reinforced-concrete and timber columns are given in Figs. 8 to 10 (p. 26–29), and further details of protection are given in Tables 3*i* and 3*j* (p. 48–49).

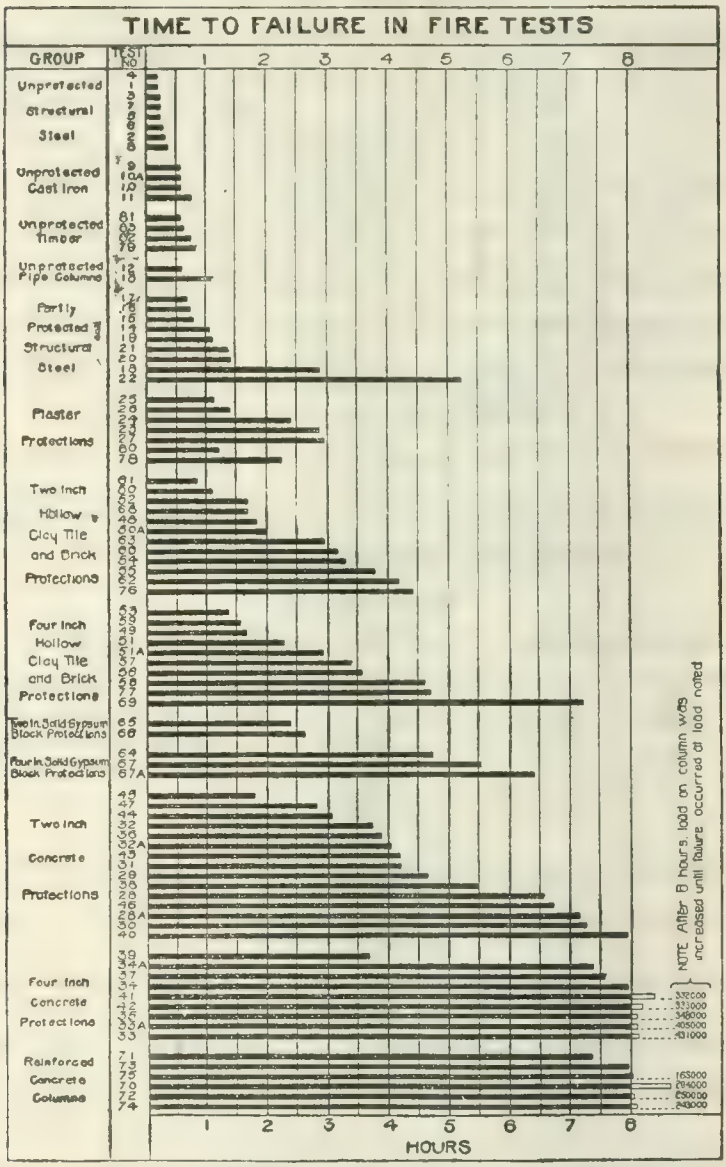


FIG. 42.—*Time to failure of columns in fire test series*

Since a large portion of the effects shown on the photographs of columns after test is due to the deformation and deflection taking place at failure, they should not be considered as representing the condition of the column or its covering immediately before this point was reached. The general condition of the test columns near the end of the fire period is indicated in their respective test logs.

3. FURNACE TEMPERATURES

The temperatures of the furnace in the fire tests, as indicated by thermocouples located on two levels at symmetrical points within the chamber, are given by the upper curves in Figs. 90 to 141, Appendix B (pp. 249–302).

(*a*) VARIATIONS FROM AVERAGE CURVE

To obtain a measure of the variation in furnace exposure between the different tests, the area under the average furnace curve given in Fig. 39, was calculated up to the end of successive intervals and compared with the area under the average furnace curve for each test up to a point near failure. The comparisons are given in Tables 42*a* to 42*i* as percentages of the area under the average curve of all tests.

4. COLUMN TEMPERATURES

The temperatures attained in the test columns and their coverings are given by the lower curves in Figs. 90 to 141, Appendix B. The arrows on the plots indicate the time of failure for each test.

(*a*) TEMPERATURE VARIATION OVER LENGTH OF COLUMN

The highest temperature generally obtained over the middle half of the column, the loss of heat at the ends due to conduction tending to maintain a lower temperature in the adjacent portions of the column. The difference was seldom very large, and in some tests, due to local variations in furnace temperature or in column protection, the highest temperature was indicated by one of the end couples.

(*b*) TEMPERATURE VARIATION OVER CROSS SECTION

The temperature variation laterally across the section was quite marked in all protected-column tests, the highest temperature in the metal obtaining almost invariably at the outer edges. In Fig. 43 this variation is given for some typical columns near failure.

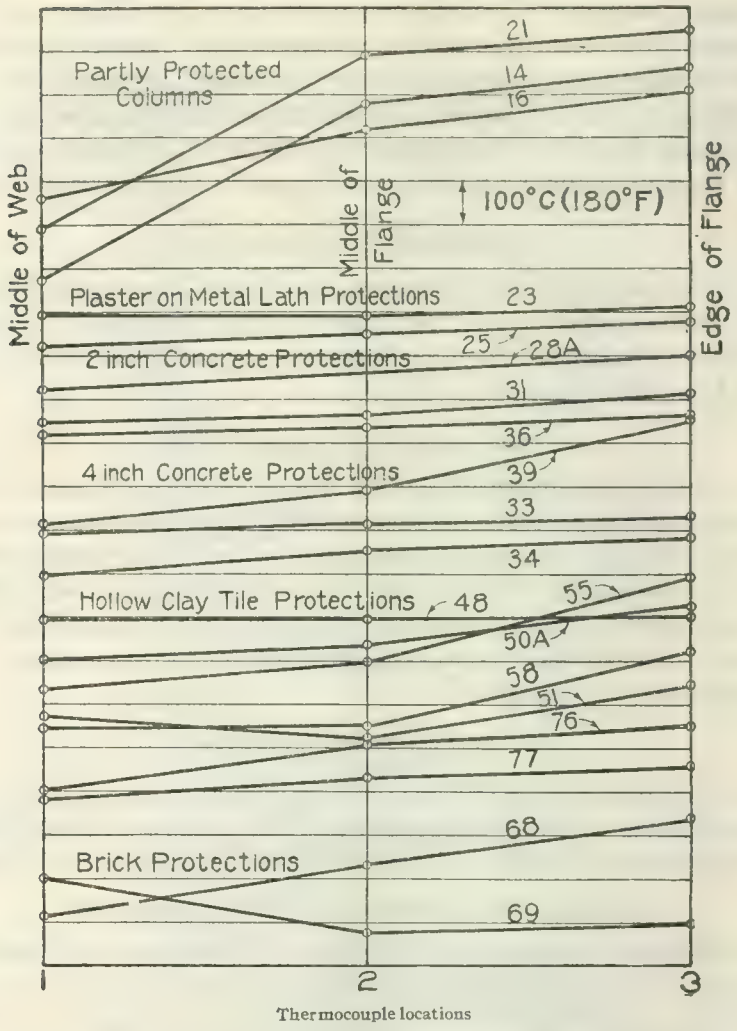


FIG. 43.—*Temperature variation over cross-section of typical columns*

The temperature indicated by the couple in the covering varied with its distance from the surface and the character of the covering material.

(c) DEHYDRATION POINTS

In the tests with concrete or gypsum protections and reinforced-concrete columns, temperatures in the column of about 100° C obtained over a considerable length of time, due to evaporation of water in the concrete and gypsum. To less extent this effect was present in tests with plaster on metal-lath and concrete-filled hollow clay-tile protections.

5. LONGITUDINAL DEFORMATION AND AVERAGE TEMPERATURE

The unit longitudinal deformation measured over a 37-inch gage length located immediately above the midheight of the column (Fig. 33), together with the computed average effective temperature over the same length, are plotted for a number of fire tests in Figs. 146 to 171, Appendix C (pp. 307–333).

(*a*) COMPUTATION OF AVERAGE EFFECTIVE TEMPERATURE

The relative location of gage points and thermocouple points and the temperature variation between the latter assumed in calculating the average effective temperature in the gage length, are shown in Fig. 44 for the case of the rolled H section. It can

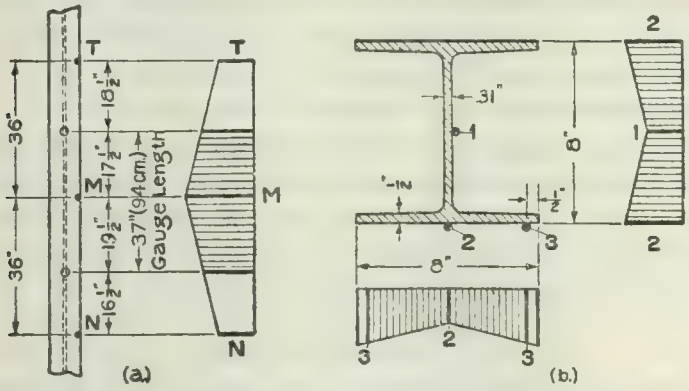


FIG. 44.—*Assumed temperature variation between thermocouple points*

be readily shown with reference to the diagram at (*a*) that the average temperature in the gage length of the edge couples, N, M, and T, located in the (3) position, is given by

$$0.115\ T + 0.143\ N + 0.742\ M = \text{Average (3)},$$

where T, N, and M are the temperatures indicated by the respective thermocouples at a given time. Similarly with reference to the diagram at (*b*) it can be shown that the average temperature in the cross section at a given level, say at the M position, is expressed by,

$$0.17\ (1) + 0.38\ (2) + 0.45\ (3) = \text{Average } M,$$

(1), (2), and (3) being the temperatures indicated by thermocouples in the respective positions in the cross section. The average effective temperature in the gage length (Av. $G\ L$) can then be obtained from the relation,

$$\text{Average } GL = \left(\frac{\text{Average } 3}{M\ 3}\right) \text{Average } M.$$

For the other structural-steel sections the average temperature at the M or N level, was obtained by taking the sum of parts of temperature readings of couples in the (1), (2), and (3) positions as given below, the derivation being similar to that for the rolled H section:

Plate and angle 0.33 (1) +0.38 (2) +0.29 (3)
Plate and channel 0.20 (1) +0.27 (2) +0.53 (3)
Latticed channel 0.27 (1) +0.365 (2) +0.365 (3)
Z bar and plate 0.441 (1) +0.232 (2) +0.327 (3)
I beam and channel 0.223 (1) +0.294 (2) +0.483 (3)
Latticed angle, test No. 26 0.484 (1) +0.35 (2) +0.166 (3)
Latticed angle, test No. 60 0.513 (2) +0.487 (3)
Starred angle, 0.416 (1) +0.584 (2)

The average temperature in the vertical plane was obtained from the same expression as that given for the rolled H section in all cases except that for the plate and channel section, due to a slight difference in the relative positions of couples and gage points, the average temperature was given by—

$$0.122\ T + 0.742\ M + 0.136\ N.$$

In the case of the cast-iron and pipe columns the average temperatures plotted pertain to the outer surface of the metal. Readings obtained on the inside of a number of the unfilled cast-iron columns indicate temperatures generally lower by 5 to 10° C. For the reinforced-concrete columns, Figs. 169 to 171, the temperatures plotted are those of the vertical reinforcing bars.

The maximum error involved in computing average temperatures by the above method is estimated to be within 20° C for tests where the covering material remained in place until near the time of failure. Where parts of the covering fell off, as for tests plotted in Figs. 163, 165, and 166, the resulting local heating and irregular temperature distribution introduced much larger errors, as evidenced by the discrepancy between the computed temperature and the corresponding unit deformation. For such tests the limit of error is probably as high as 50° C.

(*b*) DEFORMATION UNDER HEAT AND LOAD

On application of working load to the column before starting the fire test, a compressive deformation resulted, which in a few tests increased slightly during the first part of the fire period, caused either by shrinkage stresses set up by dehydration of the covering or by increased load on the metal portion of the column resulting from cracking of the covering. As the temperature increased the steel, cast-iron, and reinforced-concrete columns expanded up to a point where the yielding due to the load became equal to or greater than the thermal expansion. (See curves, pp. 308–333).

The unit expansion per degree C during this period varied generally between 0.000010 and 0.000014 for the steel and cast-iron columns, the average of the calculated values for each of these two types of columns being very nearly 0.0000125. These were taken from room temperature up to a point where the rate of expansion began to decrease rapidly due to yielding of the metal. The total observed unit expansion up to the point where expansion ceased varied in the tests of steel columns from 0.0044 to 0.0066, the average being 0.0054. For the cast-iron columns the variation was from 0.0060 to 0.0071, with an average of 0.0064. The lower values are due mainly to local heating which caused the metal to yield over a relatively short portion of the gage length before full expansion had been attained elsewhere. The columns under the lower unit loads generally attained higher expansion than those more heavily loaded.

In the tests of reinforced-concrete columns unit expansions of 0.0000095 to 0.0000102 per degree C obtained, with values of maximum unit expansions of 0.0023 to 0.0046. The point of maximum expansion was less sharply defined than for the steel and cast-iron columns, and the expansion attained was lower.

The compressive deformation taking place subsequent to maximum expansion varied in rate and duration with the type of column and the load-carrying capacity of the covering material. In some tests the total compression before failure more than equaled the previous expansion.

In the tests of timber columns slight expansions were noted during the first few minutes of the fire period, subsequent to which the movement was one of progressive depression, the principal deformation occurring at the bearings on the steel or cast-iron cap introduced near the top of the column.

(*c*) PERIOD OF EXPANSION

The period of expansion and time to failure of all columns in the fire-test series are given in Table 43 (pp. 122–123). A comparison is given between the length of the portions of the test period preceding and following the point of maximum expansion, as also the maximum temperatures obtaining at the latter point and at failure.

The period of expansion of the steel, cast-iron, and concrete columns is shown in chart form in Fig. 45.

(*d*) MAXIMUM COLUMN TEMPERATURES

These are given in Table 43 for the point of maximum expansion and the time of failure, the temperatures being obtained from the time-temperature curves extended where necessary to the failure point in the given test.

The edge temperature for the structural-steel columns is the maximum temperature indicated by the couple located nearest to the edge of the section on any of the four regular thermocouple levels. For the cast-iron columns the temperatures given pertain to the outer surface of the metal and for the reinforced-concrete columns to the vertical reinforcing bars. The edge temperatures given for the timber columns were measured on the metal cap at the edge of the column bearing.

The maximum average temperatures for the sections of the columns having the highest edge temperatures were computed by the method given in paragraph 5*a* above, the limits of error involved being somewhat higher than for the plotted results, as each determination is based on fewer couple readings. In the case of the unprotected and partly protected columns, gypsum-block protections, and a few of the concrete and hollow clay-tile protections the average temperatures are not given, since the rapid temperature rise and irregular distribution obtaining near the end of these tests made the computed results unreliable.

At the point of maximum expansion the average over the sections having the maximum temperature varies for tests of structural steel from 484 to 593° C (903 to 1099° F), with an average of 530° C (986° F). Similarly at the time of failure in the fire tests the computed values vary for the given structural-steel columns from 570 to 837° C (1058 to 1601° F), the average being 668° C (1234° F). The high temperatures obtaining at failure in a number of the tests indicate that at this point the covering carried

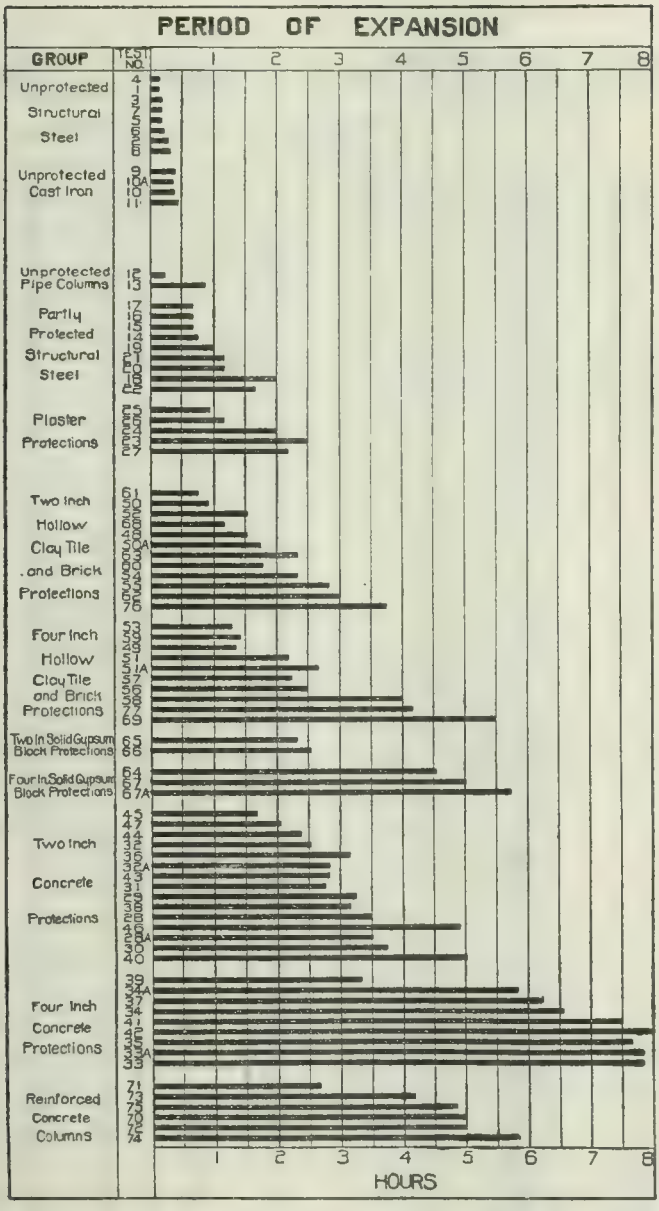


FIG. 45.—*Expansion period of steel, cast iron, and concrete columns in fire-test series*

TABLE 43.—TIME TO FAILURE, PERIOD OF EXPANSION AND MAXIMUM COLUMN TEMPERATURES

Test No.	Period of Expansion Hr.—Min.	Time to Failure, Hr.—Min.	Difference, Per cent of Expansion Period	Maximum Observed Temperature in Metal, Deg. C.					
				At End of Expansion Period			At Failure		
				Couple Level	Edge	Average	Couple Level	Edge	Average
1	0—08	0—11¼	40.6	B	530	.....	B	624	.....
2	0—17	0—19¼	13.2	N	633	.....	N	668	.....
3	0—11	0—14	27.3	M	557	.....	M	626	.....
4	0—09	0—11	22.2	M	510	.....	M	578	.....
5	0—11	0—14¼	29.5	M	620	.....	M	670	.....
6	0—13	0—17	30.8	B	623	.....	B	660	.....
7	0—11	0—14	27.3	M	547	.....	M	622	.....
8	0—19	0—21½	13.1	B	575	.....	B	620	.....
9	0—24	0—34¼	42.7	M	584		M	694	
10	0—23	0—34½	50.0	B	638	Measured	B	745	Measured
10A	0—22	0—34¼	55.7	N	626	on surface	N	718	on surface
11	0—27	0—45¼	67.6	B	570	of column	B	758	of column
12	0—14	0—36	157.0	N	502		.....		
13	0—52	1—11¾	37.9	M	781		M	872	
14	0—45	1—04¼	42.8	M	623	.....	M	810	.....
15	0—40	0—48¼	20.6	N	670	.....	N	730	.....
16	0—40	0—44½	11.2	M	671	.....	M	717	.....
17	0—40	0—41¾	4.4	N	700	.....	N	724	.....
18	2—00	2—53	44.1	N	672	.....	N	788	.....
19	1—00	1—07¼	12.1	M	831	.....	M	860	.....
20	1—10	1—24½	20.7	N	853	.....	N	909	.....
21	1—10	1—21¾	16.8	M	820	.....	M	858	.....
22	1—40	5—14	214.0	M	504	.....	N	790	.....
23	2—30	2—52	14.7	M	573	556	M	650	635
24	2—00	2—24	20.0	N	504	498	N	614	609
25	0—57	1—07¾	18.9	N	564	522	N	653	610
26	1—10	1—23½	19.3	N	512	501	N	606	598
27	2—11	2—58	35.9	T	607	.....	N	735	.....
28	3—30	6—33¾	87.5	M	590	545	N	750	716
28A	3—31	7—09¼	103.5	N	515	484	N	723	690
29	3—15	4—38½	42.8	N	565	525	N	735	693
30	3—45	7—16	93.8	N	570	535	.....		
31	2—46	4—11½	51.5	N	533	504	N	732	702
32	2—32	3—44	47.4	N	547	503	B	722	685
32A	2—50	4—02	42.4	M	576	486	N	736	656
33	7—50	*8—08	.....	B	546	533	B	*554	*541
33A	7—50	*8—07¼	.....	M	536	522	M	*547	*532
34	6—34	7—58	21.3	N	598	574	N	718	690
34A	5—50	7—23	26.6	N	571	542	N	732	700
35	7—40	*8—07	.....	N	539	527	N	*558	*546
36	3—10	3—53¼	22.8	B	603	571	B	695	664
37	6—10	7—34½	22.8	B	545	525	B	664	644
38	3—10	5—28¾	73.0	N	531	516	N	696	683
39	3—20	3—41¼	10.6	M	759	.....	M	794	.....
40	5—01	7—57	58.5	B	574	536	M	881	837
41	7—30	*8—24¼	.....	N	527	496	N	*561	*535
42	8—00	*8—11½	.....	B	553	526	B	*560	*535
43	2—50	4—11	47.6	N	601	553	N	769	724
44	2—23	3—04½	28.8	N	562	520	N	727	677
45	1—41	1—47	5.9	B	519	510	B	575	570
46	4—55	6—43½	36.8	N	543	.....	N	695	.....
47	2—03	2—48¾	37.2	N	568	.....	N	710	.....
48	1—32	1—50	19.6	M	542	537	M	647	636
49	1—21	1—40	23.4	M	561	.....	M	654	.....
50	0—55	1—05¾	19.5	N	682	.....	N	775	.....
50A	1—44	1—59½	14.9	B	695	.....	B	778	.....
51	2—11	2—17¼	4.8	N	617	546	N	670	601
51A	2—40	2—55½	9.7	N	600	549	N	678	622
52	1—32	1—40¾	9.5	N	520	497	N	593	570
53	1—17	1—22¼	6.8	N	580	535	N	659	603
54	2—20	3—17¼	40.9	B	543	493	B	820	764
55	2—51	3—46¾	32.6	M	684	527	M	889	692
56	2—30	3—33½	42.3	M	643	564	M	795	732
57	2—15	3—23	50.4	N	749	593	T	894	755
58	4—00	4—35¾	14.9	T	624	523	T	714	591
59	1—25	1—33¾	10.3	.....			.....		
60	1—47	3—09½	77.0	N	561	532	N	750	737
61	0—45	0—50¼	11.7	N	585	571	N	638	625
62	3—00	4—11½	39.7	M	584	.....	M	760	.....
63	2—20	2—57½	26.8	N	598	.....	N	730	.....

*Column loaded to failure after 8-hr. fire exposure.

TABLE 43.—TIME TO FAILURE, PERIOD OF EXPANSION, AND MAXIMUM COLUMN TEMPERATURES—Concluded

Test No.	Period of Expansion Hr.—Min.	Time to Failure, Hr.—Min.	Difference, Percent of Expansion Period	Maximum Observed Temperature in Metal, Deg. C.					
				At End of Expansion Period			At Failure		
				Couple Level	Edge	Average	Couple Level	Edge	Average
76	3—45	4—25½	18.0	N	553	524	N	663	619
77	4—10	4—42¼	12.9	N	600	558	B	690	652
64	4—32	4—43¼	4.1	T	513	.....	T	868	.....
65	2—20	2—21½	1.1	N	479	.....	N	490	.....
66	2—32	2—36	2.6	T	463	.....	T	491	.....
67	5—01	5—31½	10.1	N	496	.....	.....	.....	.....
67A	5—45	6—24½	11.4	.....	.....	.....	M	980	.....
68	1—10	1—40¾	43.9	N	606	516	N	754	677
69	5—30	7—13¼	31.3	N	550	553	N	706	716
70	5—00	*8—40¼	.....	N	642	Measured on vertical reinforcing bars	N	*797	Measured on vertical reinforcing bars
71	2—40	7—22¾	176.6	T	468		B	942	
72	5—00	*8—04½	.....	T	475		T	*623	
73	4—10	7—57½	91.0	N	528		N	845	
74	5—50	*8—06½	.....	B	493		N	*589	
75	4—50	*8—01¾	.....	B	552		B	*788	
78	.....	2—15¼	.....	.....	.....	.....	H	360	.....
79	.....	0—50	.....	.....	.....	.....	H	510	.....
80	.....	1—13	.....	.....	.....	.....	H	564	.....
81	.....	0—35	.....	.....	.....	.....	H	432	.....
82	.....	0—45¼	.....	.....	.....	.....	H	510	.....
83	.....	0—38½	.....	.....	.....	.....	H	544	.....

*Column loaded to failure after 8-hr. fire exposure. H. temperature measured on metal cap at edge of timber column bearing.

a large proportion of the applied load. The same effect appears to have influenced to much smaller extent the temperatures attained at the point of maximum expansion.

The surface temperatures at failure and at maximum expansion in the tests of cast-iron columns were on the average about 70° C (126° F) higher than the average over the section for structural-steel columns at the given stages of the test.

6. TOTAL VERTICAL DEFORMATION

(*a*) BEFORE FAILURE

Measurement of the movement of the head of the loading ram was made for a number of columns as described in Sec. VIII, paragraph 4, and the results are plotted in Figs. 46 and 47. An approximate measure of the total expansion and depression was also obtained from the card of the indicator (Sec. VI, par. 3*b*) mounted on the control board.

The total expansion of the steel columns varied from ⅞ to ⅛ inch and of the cast-iron columns from 1 to ½ inch, the lower values being due to local heating caused by impairment of the covering over a short length, which induced failure while other

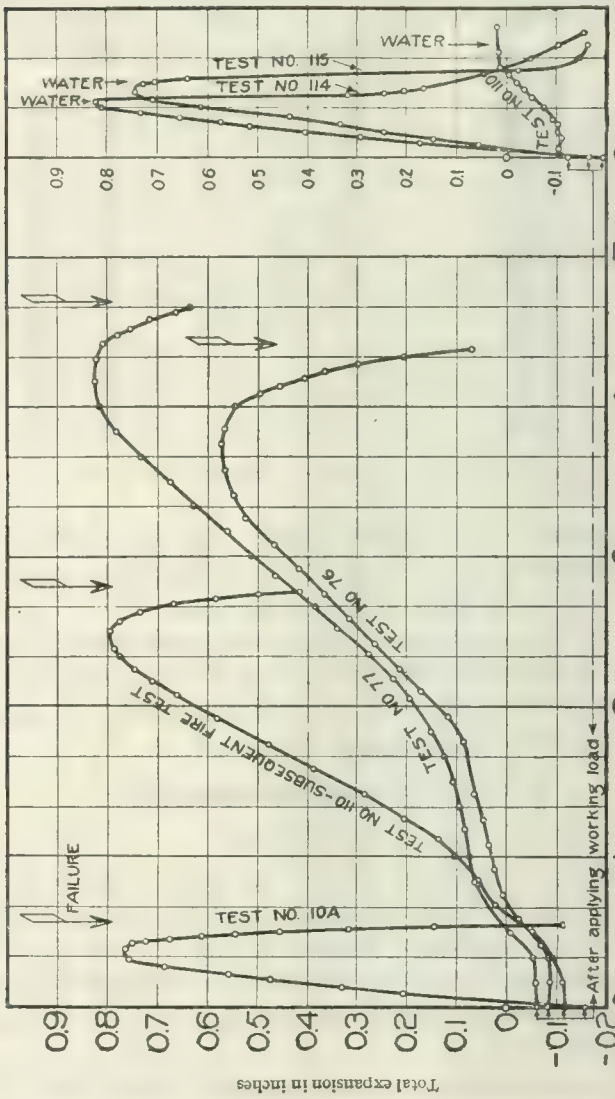


FIG. 46.—*Total expansion, test Nos. 10A, 76, 77, 110, 114, and 115*

portions of the column were at much lower temperature. The expansion of the reinforced-concrete columns was about one-half that of the cast-iron columns.

The depression of the top of the timber columns is given in Fig. 47. These deformations were due mainly to crushing of the wood at the metal cap, the heat conducted by it into the bearing causing a large reduction in the compressive strength of the wood in contact with it.

(*b*) AT FAILURE

With the setting used for the cut-off valve of the cylinder, the resulting downward movement at failure varied between 2½ and 3¼ inches. As given on the indicator card, from ¼ to 2½ inches of this travel was made under nearly full pressure. The indicated pressure in the cylinder at the end of the travel varied from ¼ to ½ of the original pressure in tests with steel columns and some of the cast-iron columns. In the case of the cast-iron columns that broke before the end of the travel and of the reinforced-concrete columns that failed under working load, the pressure indicated immediately before the valve cut off was less than ¼ of the original pressure.

7. LATERAL DEFLECTION

(*a*) BEFORE FAILURE

The center deflections observed, where large enough to have any bearing on the manner of failure of the column, are noted in the respective test logs. In most tests, decided deflection did not begin until after the point of maximum expansion was passed. The deflection observed at the last reading before failure, varied for different tests from less than ⅛ to 2½ inches. In almost all cases the direction of the deflection before failure was the same as that of the final buckle. Initial general bends up to $\frac{3}{16}$ of an inch appear to have had no influence on the direction of the deflection of the column either before or at failure.

The maximum measured deflections of the reinforced-concrete columns before the end of the 8-hour fire period varied from less than ⅛ to ¼ of an inch.

(*b*) DEFLECTION AT FAILURE

The maximum deflections at failure of the steel columns as measured after they were taken out of the furnace, varied from

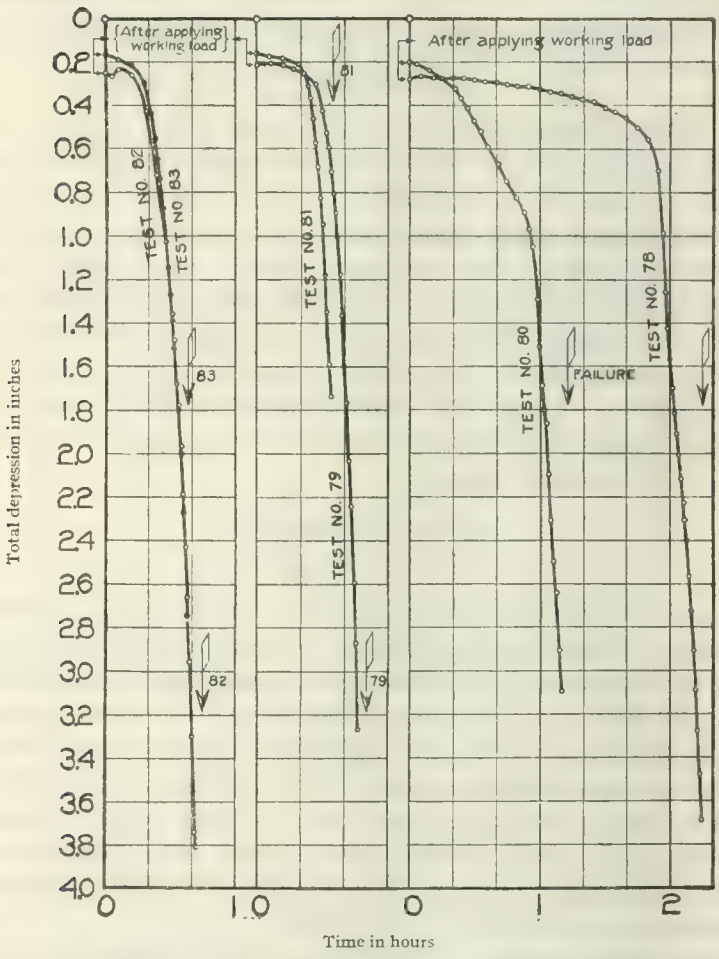


FIG. 47.—*Depression of top of timber columns*

less than 1 to 15⅜ inches. The smaller deflections obtained in tests where the failure was due to local buckling of individual section members or by direct crushing. The sections that generally failed locally were the plate and channel section and the latticed sections, although the other sections developed this type of failure when exposed to local heating. The large deflections were coincident with more nearly uniform heating and failure with deflection of the column as a whole. The direction of the deflection conformed quite uniformly with the line of least rigidity of the section. The strains at failure caused the thinner plates and section members to buckle between rivet points. The rivets were seldom found to have sheared.

The cast-iron columns usually failed by breaking at one or more points. The curvature of the pieces indicated that deflections of over 9 inches had been sustained before fracture occurred. In a few cases loading was discontinued after the column failed to sustain working load and before fracture occurred, with resultant deflections of 3 to 5 inches.

The failure of the reinforced-concrete columns was due to local crushing and shearing and developed little deflection of the column as a whole.

The failure of the timber columns with the steel plate cap and timber strut bearing was accompanied by lateral slipping of the cap on its bearings due to the softening of the wood immediately adjacent to them. In the case of the timber columns with cast-iron cap and pintle bearing, the immediate cause of failure was fracture of the cap. In both cases the deformation in the wood at the cap, before slipping or fracture of the latter, was so large as to constitute a near equivalent of the ensuing failure.

8. LOG OF FIRE TESTS

The log of the tests includes visual observations during the test, the time of maximum expansion, the time of failure, and effects at failure. The latter was usually accompanied by general or local buckling, the point of maximum lateral deflection being in or near the most highly heated region of the column.

The references after each test give in order the figure number of the corresponding column views, time-temperature curves, the deformation and average temperature curves.

(*a*) UNPROTECTED COLUMNS

TEST NO. 1. ROLLED H.—7 mins.—trace of color on flange edges. 8 mins.—maximum expansion. Lateral deflection at middle of column, 1/8 in. to west, increasing to 3/4 in. at 10 1/2 mins. 11 1/4 mins.—failure with buckling to west, maximum at 4 1/4 ft. above base. (Figs. 58, 90, and 146.)

TEST NO. 2. PLATE AND ANGLE.—6 mins.—trace of color on flange edges increasing to cherry red at 15 mins. 17 mins.—maximum expansion. 16 mins.—deflection 1/8 in. to east, increasing to 1/4 in. at 18 1/2 mins. 19 1/4 mins.—failure with buckling to east, maximum at 5 1/2 ft. above base. (Figs. 58, 90, and 146.)

TEST NO. 3. PLATE AND CHANNEL.—10 mins.—trace of color on flanges. 11 mins.—maximum expansion. 14 mins.—failure with buckling to west, maximum at 7 1/2 ft. above base. Column low cherry red in color. (Figs. 58, 90, and 146.)

TEST NO. 4. LATTICED CHANNEL.—9 mins.—maximum expansion. 10 mins.—trace of color on flange edges and web. 11 mins.—failure with buckling to east, maximum at 3 1/2 ft. above base. (Figs. 58, 90, and 146.)

TEST NO. 5. Z BAR AND PLATE.—9 mins.—deflection of 1/8 in. north increasing to 1/4 in. north at 11 1/2 mins. 10 mins.—dull red color noted. 11 mins.—maximum expansion. 14 1/4 mins.—failure with buckling to north, maximum at 6 ft. above base. (Figs. 58, 91, and 146.)

TEST NO. 6. I BEAM AND CHANNEL.—8 mins.—no deflection. 10 mins.—trace of color on flanges. 13 mins.—maximum expansion. 16 mins.—deflection of 3/4 in. north. 17 mins.—failure with buckling to north, maximum at 5 1/2 ft. above base. (Figs. 59 and 91.)

TEST NO. 7. LATTICED ANGLE.—10 mins.—trace of color noted in lattice. 11 mins,—maximum expansion. 14 mins.—failure with local buckling in each angle from 8 to 10 ft. above base. (Figs. 59 and 91.)

TEST NO. 8. STARRED ANGLE.—13 1/2 mins.—deflection of 1/8 in. northeast, increasing to 5/8 in. north at 18 1/2 mins. 15 mins.—traces of color increasing to red at 18 mins. 19 mins.—maximum expansion. 21 1/2 mins.—failure with buckling to north, maximum at 5 ft. above base. (Figs. 59 and 91.)

TEST NO. 9. ROUND CAST IRON, ENDS RESTRAINED.—18 mins.—traces of color increasing to bright red at 32 min. 24 mins.—maximum expansion. 26 mins.—deflection of 3/8 in. west, increasing to 1 in. west at 31 mins. 34 1/4 mins.—failure, buckling to west and breaking at 1 1/2, 5, and 11 ft. above base. Thinnest metal on southeast side. (Figs. 60, 92, and 147.)

TEST NO. 10. ROUND CAST IRON.—20 mins.—dull red color increasing to cherry red at 28 mins. 23 mins.—maximum expansion. 26 mins.—deflection of 1/4 in. northwest, increasing to 1/2 in. west at 31 mins. 34 1/2 mins.—failure, buckling to southwest and breaking about 5 ft. above base. Thinnest metal on northeast side. (Figs. 60, 92, and 147.)

TEST NO. 10A. ROUND CAST IRON.—26 mins.—no color noted. Deflections not measured. 22 mins.—maximum expansion. 34 1/4 mins.—failure, buckling to north and breaking in two about 5 ft. above base. Column glowing red. (Figs. 60, 92, and 147.)

TEST NO. 11. ROUND CAST IRON, CONCRETE FILLED.—33 mins.—traces of color increasing to dull red at 39 mins. 27 mins.—maximum expansion. 31 mins.—deflection of 3/8 in. west, increasing to 1 1/4 in. west at 41 mins. 45 1/4 mins.—failure, buckling to west and breaking in two about 5 ft. above base. Thinnest metal on north side of section. (Figs. 60, 92, and 147.)

TEST NO. 12. STEEL PIPE, CONCRETE FILLED.—14 mins.—maximum expansion. 21 mins.—traces of color increasing to cherry red at 32 mins. 12½ mins.—deflection of ¼ in. northeast, increasing to ¾ in. northeast at 31 mins. 36 mins.—failure, buckling to northeast, maximum at 6½ ft. above base. (Figs. 60, 93, and 147.)

TEST NO. 13. REINFORCED STEEL PIPE, CONCRETE FILLED.—15 mins.—traces of color increasing to bright red at 38 mins. 52 mins.—maximum expansion. 4 mins.—slight deflection to northwest noted, increasing to ¾ in. at 29 mins., 1⅝ in., at 1 hour, and 2⅝ in. northwest at 1 hr., 8 mins. 1 hr. 11¾ mins.—failure, buckling to northwest, maximum at 6 ft. above base. (Figs. 60 and 93.)

(*b*) PARTLY PROTECTED COLUMNS

TEST NO. 14. ROLLED H, 1:2:4 JOLIET GRAVEL CONCRETE.—No cracks or spalling noted in concrete before failure. 45 mins.—maximum expansion. 25 mins.—slight deflection to west noted, increasing to ¼ in. at 46 mins. 1 hr. 4¼ mins.—failure, buckling to west, maximum at 7 ft. above base. Loud report at failure probably due to failure of concrete in compression.

After failure.—Concrete generally loose, due to large deflection at failure; no marked heat or dehydration effects. Two pieces of concrete filling 3¾ by 7$\frac{1}{16}$ by 10 in. tested on end three weeks after the fire test gave ultimate compressive strengths of 1440 and 1800 lb. per sq. in., respectively. (Figs. 61 and 94.)

TEST NO. 15. ROLLED H, 1:2:4 GRANITE CONCRETE.—25 min.—fine horizontal cracks appeared in concrete on west at center. 30 mins.—cracks on west opening up and extending, column otherwise unaffected; steel dull red. 40 mins.—maximum expansion. No spalling before failure. 24 mins.—slight deflection to west noted, increasing to ¼ in. at 36 mins. 48¼ mins.—failure, buckling to west, maximum at 7½ ft. above base.

After failure.—Concrete generally loosened, this probably occurring at failure. Piece of filling tested 9 days after the fire test had ultimate compressive strength of 960 lb. per sq. in. (Figs. 61 and 94.)

TEST NO. 16. PLATE AND ANGLE, 1:2:4 TRAP CONCRETE.—40 mins.—steel flanges dull red. Maximum expansion. No cracks or spalling noted in concrete before failure. 36 mins.—deflection of ⅛ in. to east. 44½ mins.—failure, buckling to the east, maximum at 5½ ft. above base.

After failure.—Concrete cracked and loose in several places. (Figs. 61 and 94.)

TEST NO. 17. PLATE AND ANGLE, 1:1½:4½ CINDER CONCRETE.—38 mins.—flanges of angles dull red. 40 mins.—maximum expansion. 41 mins.—no cracks or spalling of concrete. 41¾ mins.—failure, buckling to east, maximum at 5 ft. above base.

After failure.—Concrete cracked at points of maximum flexure and had fallen out to a depth of 1 in. in places. (Figs. 61 and 94.)

TEST NO. 18. LATTICED CHANNEL, 1:2:4 TRAP CONCRETE.—15 mins.—traces of color showing in concrete and all surfaces glowing at 30 min. 2 hrs.—maximum expansion. 2 hrs. 45 mins.—several vertical cracks ⅛ in. wide and about 18 in. long on east and west sides, mostly in lower half; concrete bulged out ½ to ¾ in. at cracks. 2 hrs. 53 mins.—failure, with local buckling of channels about 3½ ft. above base.

After failure.—Concrete in channels cracked along flanges in lower half and quite crumbly; concrete in upper half quite firm. A piece of concrete, filling the space between the channels near the middle of the column, 5½ by 9½ by 14 in. was tested on end 34 days after the fire test and developed an ultimate compressive strength of 577 lb. per sq. in. (Figs. 61 and 95.)

TEST NO. 19. Z BAR AND PLATE, 1 : 3 : 5 LIMESTONE CONCRETE.—20 mins.—traces of color in steel increasing to cherry red at 40 min. 50 mins.—concrete flowing. Very little cracking of concrete before failure. 1 hr.—maximum expansion. 35 mins.—deflection of ¼ in. north, increasing to 1⅛ in. at 1 hr. 4 mins. 1 hr. 7¼ mins.—failure, buckling to north, maximum at 7 ft. above base.

After failure.—Concrete generally loose from steel on north and south sides, standing out to ⅞ in. in places. A few horizontal cracks in the concrete were present on the east and west sides. A piece of filling 11¼ in. long tested 14 days after the fire test developed ultimate compressive strength of 730 lbs. per sq. in. (Figs. 62 and 96.)

TEST NO. 20. I BEAM AND CHANNEL, 1 : 3 : 5 TRAP CONCRETE.—34 mins.—trace of color on steel flanges. 36 mins.—color in concrete. 40 mins.—flanges dull red. 60 mins.—concrete and exposed steel bright red. 1 hr. 8 mins.—no cracks or spalling of concrete. 1 hr. 10 mins.—maximum expansion. 1 hr. 24½ mins.—failure, buckling to south, maximum at 6 ft. above base.

After failure.—Concrete remained in place except where crushed at middle, top, and bottom. Rivet heads probably helped to hold it. (Figs. 62 and 96.)

TEST NO. 21. I BEAM AND CHANNEL, 1:3:5 TRAP CONCRETE.—38 mins.—traces of color on steel flanges. 1 hr., 10 mins.—maximum expansion. 1 hr. 20 mins.—no cracks or spalling of concrete. Steel flanges dark red, concrete bright red. 1 hr. 21¾ mins.—failure, buckling to north, maximum at 6 ft. above base.

After failure.—Concrete remained in place except where crushed at failure. (Figs. 62 and 96.)

TEST NO. 22. LATTICED ANGLE, 1:2:4 LIMESTONE CONCRETE.—25 mins.—concrete glowing on corners; a number of fine cracks noted on all faces. 1 hr. 40 mins.—maximum expansion. 3 hrs.—surface of column glowing at white heat. 4 hrs. 50 mins.—large vertical cracks appeared under brackets near top. Very little cracking or spalling of concrete before failure. 5 hrs. 14 mins.—failure with local buckling of steel and crushing of concrete about 11 ft. above the base. Unprotected brackets probably caused failure at this point by conducting heat into angles.

After failure.—Except at point of failure concrete did not appear greatly damaged. Limestone calcined to depth of 1 in., and one month after test the whole exterior concrete had become loose due to air slaking of the lime. From temperature of steel at 3 hrs. it is probable that the concrete carried most of the load after this time. (Figs. 62 and 95.)

(c) PLASTER ON METAL-LATH PROTECTIONS

TEST NO. 23. PLATE AND ANGLE, TWO 1-IN. LAYERS OF PORTLAND-CEMENT PLASTER ON EXPANDED METAL LATH.—12½ mins.—distinct thud heard caused by failure of covering due to expansion; plaster cracked and spalled on all faces at top of column near bottom of bracket. 23 mins. to 1 hr. 15 mins.—some six or eight fine vertical and horizontal cracks, 3 to 8 in. long, appeared on east, south, and west sides from 1 to 8 ft. above base. Very little change up to failure, except that all cracks opened up slightly. 2 hrs. 30 mins.—maximum expansion. 2 hrs. 41 mins.—deflection of ⅛ in. northwest, increasing to 1 in. west at 2 hrs. 51 mins. 2 hrs. 52 mins.—failure, buckling to west, maximum at 7½ ft. above base.

After failure.—Plaster of outer layer dehydrated and both outer and inner layers very crumbly. Both layers fairly well keyed to lath. (Figs. 63, 97, and 148.)

TEST NO. 24. PLATE AND CHANNEL, TWO ⅞-IN. LAYERS OF PORTLAND-CEMENT PLASTER ON WOVEN WIRE LATH.—8½ mins.—distinct thud heard; plaster cracked and spalled on all faces at bottom of bracket exposing lath on corners. 14 mins. to 1 hr. 45 mins.—some 15 fine vertical cracks, 3 to 8 in. long, appeared on all sides near corners. 2 hrs.—maximum expansion. 2 hrs. 15 mins.—very little change except some cracks opened up to ⅛ in. 2 hrs. 24 mins.—failure, with local buckling about 6 ft. above base.

After failure.—Outer layer of plaster dehydrated; inner layer fairly hard except at failure point. Plaster well keyed to lath. (Figs. 63, 97, and 148.)

TEST NO. 25. Z BAR AND PLATE, ONE 1-IN. LAYER OF PORTLAND-CEMENT PLASTER ON EXPANDED METAL LATH.—10 mins.—slight noise heard; plaster cracked and spalled on all sides under bracket. 48 to 54 mins.—fine vertical cracks noted on all faces near corners, opening up slightly toward end of test. 57 mins.—maximum expansion. 41 mins.—slight deflection to north noted, increasing to ¼ in. at 1 hr. 1 min. 1 hr. 7¾ mins.—failure, buckling to north, maximum at 5½ ft. above base.

After failure.—Plaster appeared to be in fairly good condition and was quite hard, except where crushed. Plaster well keyed, covering inner face of lath. (Figs. 63, 98, and 149.)

TEST NO. 26. LATTICED ANGLE, ONE 1⅛-IN. LAYER OF PORTLAND-CEMENT PLASTER ON EXPANDED METAL LATH.—10 to 17 mins.—cracking and some spalling of plaster on bracket. 20 mins. to 1 hr.—some 15 fine vertical cracks, 6 to 12 in. long, appeared near corners on all sides; also horizontal cracks near bottom. Before failure bracket cracks had opened up to ½ in. and others nearly ⅛ in. 1 hr. 10 mins.—maximum expansion. 1 hr. 23½ mins.—failure, with local buckling about 6 ft. above base.

After failure.—Very little strength in plaster, very crumbly. Keys covered lath on inside. ¾ in. air space between plaster and angles. (Figs. 63, 98, and 149.)

TEST NO. 27. ROUND CAST IRON, ONE 1½-IN. LAYER OF PORTLAND-CEMENT PLASTER ON HIGH-RIBBED METAL LATH.—18 mins.—vertical and horizontal cracks under bracket. 1 hr. 40 mins. to 2 hrs. 5 mins.—several fine vertical and horizontal cracks, 3 to 12 in. long, mostly near middle. 2 hrs. 11 mins.—maximum expansion. 2 hrs. 50 mins.—cracks opening up to ⅛ in. 2 hrs. 30 mins.—slight deflection to southwest noted, increasing to ½ in. at 2 hrs. 51 mins., and 1¼ in. southwest at 2 hrs. 56 mins. 2 hrs. 58 mins.—failure, buckling to southwest and breaking about 6 ft. above base.

After failure.—Plaster fairly hard where not crushed. Plaster pushed through to iron except at ribs; average thickness of plaster 1½ in. (Figs. 63, 98, and 149.)

(*d*) CONCRETE PROTECTIONS

TEST NO. 28. ROLLED H. 2-IN. 1:2:4 LIMESTONE CONCRETE.—30 to 45 mins.—a few fine vertical cracks at middle and top of east face opening up slightly toward failure. No spalling before failure. 3 hrs. 30 mins.—maximum expansion. 6 hrs. 33¾ mins.—failure, buckling to west, maximum at 6½ ft. above base.

After failure.—Concrete fairly hard although calcined on surface. Flanges exposed at middle and flange edges exposed at several points, this all occurring at failure. Wire tie not broken. (Figs. 64 and 99.)

TEST NO. 28A. ROLLED H. 2-IN. 1:2:4 LIMESTONE CONCRETE. NOT TIED.—2 hrs. 55 mins. to 3 hrs. 23 mins.—several fine vertical cracks 3 to 12 in. long and about 3 in. from corners appeared on east and west faces. No spalling or other effects noted before failure. 3 hrs. 31 mins.—maximum expansion. 5 hrs.—slight deflection to west noted, increasing to ¼ in. at 6 hrs. 2 mins., and 1½ in. west at 7 hrs. 2 mins. 7 hrs. 9¼ mins.—failure, buckling to west, maximum 6¾ ft. above base.

After failure.—Concrete surface fairly firm although discoloration and calcination extend to depth of 1 in. Concrete fell off at failure, exposing about one-half of area of steel flanges. (Figs. 64, 100, and 150.)

TEST NO. 29. ROLLED H. 2-IN. 1:2:4 TRAP CONCRETE.—50 mins. to 3 hrs. 50 mins.—several fine vertical and horizontal cracks appeared on east and west faces, which became larger near failure. No spalling or other effects noted before failure. 3 hrs. 15 mins.—Maximum expansion. 3 hrs. 30 mins.—slight deflection to west noted, increasing to ¼ in. at 4 hrs. 2 mins., and 1½ in. west at 4 hrs. 32 mins. 4 hrs. 38½ mins.—failure, buckling to west, maximum at 5½ ft. above base.

After failure.—Surface of concrete reddish in color. No fusion noted. Concrete fell off at failure exposing flange for 1½ ft. on south side and flange edges at other points. Wire tie not broken. (Figs. 64, 101, and 151.)

TEST NO. 30. ROLLED H. 2-IN. 1:2:4 JOLIET GRAVEL CONCRETE.—37 mins.—fine cracks noted in brackets. 3 hrs. 4 mins. to 5 hrs. 5 mins.—a number of fine cracks, 2 to 12 in. long, mostly vertical and near edges, appeared on east and west faces opening up to $\frac{1}{16}$ of an in. near end of test in some cases. No spalling before failure. 3 hrs. 45 mins.—maximum expansion. 6 hrs. 40 mins.—slight deflection to east noted, increasing to ⅜ in. at 7 hrs. 9 mins. 7 hrs. 16 mins.—failure, buckling to east, maximum at 5½ ft. above base.

After failure.—Concrete on surface fairly hard after test, but disintegrated in 30 days due to air slaking. Flanges and flange edges exposed in places due to spalling of concrete at failure. (Figs. 64 and 102.)

TEST NO. 31. ROLLED H. 2-IN. 1:2:4 SANDSTONE CONCRETE.—28 mins. to 1 hr. 5 mins.—vertical cracks appeared on east and west faces about 3 in. from edges, running nearly full length of column, opening up in places to $\frac{1}{8}$ in,; also a few fine vertical cracks on north and south faces in lower half. 45 mins.—southeast corner spalled off 3 in. deep from 3 to 6 ft. up, exposing edge of steel. 1 hr. 5 mins. to 3 hrs.—vertical cracks at corners became continuous, opening up to $\frac{3}{16}$ in.; concrete on southeast and southwest corners generally spalled off or loose in lower half, exposing edges of flanges in places. Very little change to failure except for minor cracking and spalling. 2 hrs. 46 mins.—maximum expansion. 2 hrs.—slight center deflection to northwest noted, increasing to ⅝ in. at 3 hrs. 1 min., and 2¼ in. northwest at 4 hrs. 11 mins. 4 hrs. 11½ mins.—failure, buckling to west, maximum at 5½ ft. above base.

After failure.—Concrete on flanges disintegrated and crumbly. Nearer the web the concrete was harder but had apparently lost much strength. (Figs. 64, 103, and 152.)

TEST NO. 32. ROLLED H. 2-IN. 1:2:5 CINDER CONCRETE.—3 hrs.—no cracking or spalling noted. 3 hrs. 20 mins. to 3 hrs. 38 mins.—a few small vertical cracks 4 to 6 in. long appeared near corners on east and west faces in lower half extending in length and opening up toward end of test; no spalling before failure. 2 hrs. 32 mins.—maximum expansion. 3 hrs.—slight deflection to east noted, increasing to ½ in. at 3 hrs. 31 mins., and 1⅝ in. east at 3 hrs. 43 mins. 3 hrs. 44 mins.—failure, buckling to east about 6 ft. above base.

After failure.—Concrete very crumbly to a depth of about 1 in. Quite hard and apparently little affected at column web 5 in. from surface. (Figs. 65, 104, and 153.)

TEST NO. 32A. Z BAR AND PLATE. 2-IN. 1:2:5 CINDER CONCRETE.—2 hrs. 50 mins.—maximum expansion. 3 hrs.—no cracking or spalling noted. 3 hrs. 50 mins.—fine vertical cracks developing on all faces mostly near northeast and southwest corners. 3 hrs. 30 mins.—slight deflection to north noted, increasing to ⅜ in. at 3 hrs. 51 mins., and 1⅝ in. north at 4 hrs. 1 min. 4 hrs. 2 mins.—failure, buckling to north, maximum at 5½ ft. above base.

After failure.—Concrete crumbly and dehydrated to a depth of about 1¼ in. Beyond this it was harder and apparently little affected. (Figs. 65, 104, and 153.)

TEST NO. 33. ROLLED H. 4-IN. 1:2:4 LIMESTONE CONCRETE.—2 hrs. to 4 hrs.—fine vertical cracks 6 to 12 in. long on east and west faces, 4 in. from corners, in lower half; also small cracks in bracket; cracks opening slightly toward end of test; no spalling. 7 hrs. 50 mins.—maximum expansion. 8 hrs.—column still supporting working load, with no apparent change; less than ⅛ in. deflection. 8 hrs. 2 mins.—load increased, with fire going until failure occurred under load of 431 000 lb., with local buckling about 10 ft. above base, at 8 hrs. 8 mins.

After failure.—Outer ¾ in. of concrete soft and could be easily knocked off 8 days after test, and was dehydrated for another 1½ in., inside of which it was hard and apparently little affected by the heat. (Figs. 65, 105, and 154.)

TEST NO. 33A. ROLLED H. 4-IN. 1:2:4 LIMESTONE CONCRETE. NOT TIED.—30 mins.—slight surface flaking at several points. 34 to 40 mins.—fine cracks noted on east and west sides of bracket. 2 hrs. 10 mins. to 5 hrs. 20 mins.—three or four fine vertical cracks 3 to 24 in. long on east and west faces, 3 in. from corners, in lower half; no spalling or further cracking before failure. 7 hrs. 50 mins.—maximum expansion. 8 hrs.—column still supporting load with no apparent change; less than ⅛ in. deflection. 8 hrs. 5 mins.—load increased with fire going until failure occurred under load of 405 000 lb. with buckling to the east, maximum 7 ft. above base, at 8 hrs. 7¼ min.

After failure.—In lower 8 ft. concrete checked and pitted with small holes of $\frac{1}{16}$ to $\frac{3}{16}$ in. diameter, and hard to depth of over $\frac{1}{16}$ in. Concrete calcined to depth of 1½ in. from surface. Concrete on flanges broke loose at failure due, in part, to absence of wire tie. (Figs. 65 and 106.)

TEST NO. 34. ROLLED H. 4-IN. 1:2:4 GRANITE CONCRETE.—32 mins. to 1 hr. 5 mins.—fine vertical cracks about 3 in. from corners mostly on east and south sides, 5 to 9 ft. up; also cracks on all sides in bracket. 1 hr. 28 mins.—cracks opened generally $\frac{1}{16}$ to $\frac{3}{16}$ in. width. 2 hrs. 13 mins.—crack through concrete at southeast corner 6 ft. up. 4 hrs. 7 mins.—¼-in. crack through northeast corner 6 ft. up. 6 hrs. 34 mins.—maximum expansion. 6 hrs. 40 mins.—very little change except for slight spalling on northwest corner 8 ft. up. 7 hrs. 33 mins.—spalling, northwest corner, 6 by 3 by 3 in. at 8 ft. up; spalling, southeast corner, 6 by 6 by 20 in. 6½ ft. up, exposing steel. 3 hrs.—slight deflection to east noted, increasing to ⅜ in. at 7 hrs. 31 mins. 7 hrs. 58 mins.—failure, buckling to east, maximum at 7 ft. above base.

After failure.—Incipient fusion of concrete 1½ to 2 in. in depth throughout. Concrete underneath very soft and crumbly. (Figs. 65, 107, and 155.)

TEST NO. 34A. ROLLED H. 4-IN. 1:2:4 GRANITE CONCRETE.—55 mins. to 1 hr. 45 mins.—small vertical cracks appeared on all faces 3 to 5 in. from corners, mostly from 5 to 7 ft. up, opening up to ⅛ in. in some cases; also several cracks in bracket. 3 hrs. 16 mins.—cracks opening up to ¼ in., no spalling except small piece at bracket. 4 hrs. 44 mins.—gas shut off for 1½ mins. to cool furnace. 5 hrs. 50 mins.—maximum expansion. 6 hrs. 30 mins.—gas shut off for 2 mins. to clear furnace; small spall noted on southwest corner 7½ ft. up. 7 hrs.—gas shut off for 2 mins.; incipient fusion apparently present. Minor spalling of corners noted. 6 hrs. 30 mins.—slight deflection to west noted, increasing to ½ in. at 7 hrs. 21 mins. 7 hrs. 23 mins.—failure, buckling to west, maximum at 5 ft. above base.

After failure.—Incipient fusion of concrete 1 to 1½ in. in depth over whole surface. (Figs. 65 and 108.)

TEST NO. 35. ROLLED H. 4-IN. 1:3:5 LIMESTONE CONCRETE.—56 mins.—fine vertical crack, 12 ins. long, on both faces at southeast corner at bottom, 4 in. from edge. 3 hrs.—no spalling or further cracking noted. 4 to 7 hrs.—furnace filled with heavy flame; impossible to see column. 7 hrs. 40 mins.—maximum expansion. 8 hrs.—column still supporting working load with no apparent change; deflection less than ⅛ in.; load increased with fire going until failure occurred under load of 348 000 lbs. with local buckling 7½ ft. above base, at 8 hrs. 7 mins.

After failure.—Surface concrete hard and sand fused throughout to depth of ⅛ in. to ¼ in.; limestone completely calcined up to 1 in. from surface, and soft and dehydrated up to 2½ ins. from surface. Concrete was harder farther in, although less so than in 33 and 33A. (Figs. 66, 109, and 156.)

TEST NO. 36. PLATE AND ANGLE. 2-IN. 1:2:4 TRAP CONCRETE.—37 mins.—small crack on west face, 5 in. long, running up diagonally from southwest corner, 2½ ft. up. 1 hr. 13 mins.—small spall, 3 by ¼ in. on southeast corner, 6½ ft. up. 1 hr. 20 mins. to 3 hrs. 40 mins.—several fine cracks at bracket. 3 hrs. 10 mins.—maximum expansion. 3 hrs. 46 mins.—several vertical cracks 2 in. from corners on east and west faces, 2 ft. to 4 ft. up; no spalling before failure. 3 hrs.—slight deflection to northwest noted, increasing to ⅞ in. at 3 hrs. 44 mins. 3 hrs. 53¼ mins.—failure, buckling to west, maximum at 6 ft. above base.

After failure.—Concrete soft and crumbly to a depth of 1½ to 2 in. (Figs. 66, 103, and 151.)

TEST NO. 37. PLATE AND ANGLE. 4-IN. 1:2:4 TRAP CONCRETE, ROUND SECTION.—35 mins. to 1 hr.—small cracks noted on all sides at bracket. 1 hr. 25 mins. to 1 hr. 40 mins.—spalling of southwest and southeast corners at bracket. 4 hrs. 20 mins.—no further cracking or spalling. 6 hrs. 10 mins.—maximum expansion. 6 hrs. 24 mins.—spalling across west face at bracket about 1 in. deep. 7 hrs. 10 mins.—spalling south face at bracket; impossible to observe lower part of column on account of furnace gases. 6 hrs. 30 mins.—slight deflection to west noted, increasing to ½ in. at 7 hrs. and 1¾ in. west at 7 hrs. 30 mins. 7 hrs. 34½ mins.—failure, buckling to west, maximum at 5½ ft. above base.

After failure.—Considerable fusion and flowing of concrete to depth of 2 in. and incipient fusion for 1 to 1½ in. farther in, in lower 8 ft. Partial fusion to ½ in. depth above this point. Concrete crumbly for 1 in. inside fused portion; fairly hard farther in. Trap rock extends to surface quite generally in upper portion which was not fused. (Figs. 66, 110, and 157.)

TEST NO. 38. PLATE AND CHANNEL. 2-IN. 1:2:4 JOLIET GRAVEL CONCRETE.—45 mins. to 2 hrs. 55 mins.—several fine vertical cracks appeared on east and west faces about 2½ in. from corners, 5 to 8 ft. above base, some opening up to $\frac{1}{16}$ in., also several cracks in bracket. 3 hrs. 10 mins.—maximum expansion. 3 hrs. 26 mins.—horizontal cracks on north face 4 ft. up, ¼ in. wide. 4 hrs. 6 mins.—vertical cracks on east and west faces opening up to ⅛ in.; very few cracks on north and south faces. 5 hrs. 4 mins.—concrete loose on corners 3 ft. up. 5 hrs. 28 mins.—no spalling. 4 hrs.—slight deflection to east noted, increasing to ½ in. at 5 hrs. 21 mins. 5 hrs. 28¾ mins.—failure, buckling to east, maximum at 2½ ft. above base.

After failure.—Concrete calcined to depth of ¾ in. and very crumbly to depth of 2 in. Inside of this point on web side concrete was quite hard and apparently little affected by the heat. (Figs. 66, 111, and 158.)

TEST NO. 39. PLATE AND CHANNEL. 4-IN. 1:2:4 MERAMEC RIVER GRAVEL CONCRETE.—19 to 26 mins.—vertical cracks developed on east and west faces near corners in lower half and at bracket. 26 mins.—2-ft. spall on northwest corner near middle. 28 mins. to 40 mins.—vertical cracks extending upward and opening up. 40 mins.—horizontal crack, 5 ft. up, across north face. 45 mins.—large spall on southwest corner, 7 ft. up. 1 hr. 20 mins.—cracks opening up to ½ in. in some cases. 1 hr. 26 mins.—concrete cracked through northeast corner, this corner spalling off from 4 to 8 ft. up at 2 hrs. 6 mins., exposing edge of steel for 4 ft. 2 hrs. 13 mins.—southwest corner spalled off for 3 ft. near middle exposing steel. 3 hrs.—northwest corner and west side spalled off above middle of column exposing steel on corner for 2 ft. 3 hrs. 20 mins.—maximum expansion. 3 hrs. 41¼ mins.—failure with local buckling 6½ to 8 ft. above base.

After failure.—Concrete inside of spalled or cracked portions was quite hard and split with fracture of aggregate. Aggregate also fractured by cracks produced in the fire test. (Figs. 66 and 112.)

TEST NO. 40. LATTICED CHANNEL. 2-IN. 1:2:4 TRAP CONCRETE, ROUND SECTION.—5 hrs. 1 min.—maximum expansion. 6 hrs. 56 mins.—no cracking, spalling, or fusion of concrete. 7 hrs. 25 mins.—a number of vertical cracks on east and west sides in lower half, varying from very fine to one ⅛ by 16 in. at southeast, 3 ft. up. No spalling or fusion. 7 hrs. 55 mins.—cracks opening up; the crack noted above open ¼ in.; no spalling, possibly some fusion. 7 hrs. 57 mins.—failure with local buckling about 5 ft. above base.

After failure.—Fusion of concrete from depth of ½ to 1½ in. near failure point. Very little concrete had run. Concrete outside of steel generally disintegrated; between channels it is harder. (Figs. 67, 113, and 159.)

TEST NO. 41. Z BAR AND PLATE. 4-IN. 1:3:5 LIMESTONE CONCRETE.—10 mins.—fine vertical crack, 2 in. long, on west face near north corner 7 ft. up. 1 hr. to 1 hr. 50 mins.—several vertical cracks, 3 to 12 in. long, appeared on west face near corners in middle section of column; some opening up to ⅛ in.; no spalling. 2 hrs. 50 mins.—concrete cracked through for 2-in. length on northwest corner, 5 ft. up. 7 hrs. 30 mins.—maximum expansion; lateral deflection at middle of column less than ⅛ in. 8 hrs.—column still supporting working load with little apparent change; no spalling; load increased with fire going until failure occurred under load of 332 000 lbs., buckling to south, maximum 6 ft. above base, at 8 hrs. 24¼ mins.

After failure.—Concrete reddish in color and dehydrated to depth of 1 in. from surface. Concrete outside of flanges dehydrated and crumbly. (Figs. 67, 114, and 160.)

TEST NO. 42. Z BAR AND PLATE. 4-IN. 1:3:5 LIMESTONE CONCRETE.—1 hr. 15 mins.—small crack on southwest corner at bracket. 2 hrs. 20 mins.—fine vertical crack on west face 4 in. from northwest corner, 4 ft. up. 8 hrs.—maximum expansion; column still supporting working load with little apparent change; no spalling; deflection less than ⅛ in.; load increased with fire going until failure occurred under load of 333 000 lb. with buckling to north, maximum 6 ft. above base, at 8 hrs. 11½ mins.

After failure.—Concrete affected same as in test No. 41. (Figs. 67, 115, and 161.)

TEST NO. 43. I BEAM AND CHANNEL. 2-IN. 1:2:4 SANDSTONE CONCRETE.—13 mins.—vertical crack, 4 in. long, on southeast corner, 5½ ft. up, causing slight spalling at 20 mins. 21 to 30 mins.—vertical cracks appeared on east and west faces near corners, mostly in middle part of column. 30 mins.—southeast corner spalled off to a height of 5½ ft., exposing edge of steel and ties for 4 ft. 33 to 41 mins.—vertical cracks on east and west faces extended nearly full length of column. 45 mins.—southwest corner spalled off 4 to 10 ft. up. 51 mins.—southeast corner spalled off 6 to 9 ft. up. 55 mins.—cracks opening up to ¼ in. in places. 1 hr. 15 mins.—southwest corner spalled off, lower 4 ft. 2 hrs. 50 mins.—maximum expansion. 3 hrs. 5 mins.—concrete checking all over surface. 4 hrs. 11 mins.—failure, buckling to north, maximum 6 ft. above base.

After failure.—Channel flanges and ties on south oxidized. Concrete discolored on surface and very crumbly for depth of 1 in., the rest dehydrated and possessed little strength. Fractures split aggregate, which also seems to have lost much of its strength, crumbling under light hammer blows. (Figs. 67, 116, and 152.)

TEST NO. 44. I BEAM AND CHANNEL. 2-IN. 1:3:5 SANDSTONE CONCRETE.—17 mins.—vertical crack on northeast corner, 2 to 5 ft. up, corner spalling off at 18 mins., also vertical crack on northeast corner, 3 to 7 ft. up, corner spalling off at 25 mins. 22 mins.—vertical cracks on both sides of northwest and southwest corners. 32 mins.—southeast corner spalled off in lower 3 ft. 35 mins.—vertical cracks on all faces about 2 in. from corners running full length of column. Edges of steel exposed for lower 7 ft. on northeast and southeast corners; concrete loose on northwest and southwest corners. 36½ mins.—southwest corner spalled from 6 to 7 ft. up. 1 hr. 2 mins.—spall east side near middle for length of 3 ft., exposing flange edges. 1 hr. 23 mins.—

all corners at bracket spalled or cracked. 1 hr. 24 mins.—northwest corner spalled from 6 to 9 ft. up. 1 hr. 35 mins.—southeast flange of channel dull red at middle of column, no other change. 2 hrs. 23 mins.—maximum expansion. 3 hrs. 4¼ mins.—failure, buckling to north, maximum 6 ft. above base.

After failure.—Concrete generally very soft and crumbly. Falls off column readily in large pieces. (Figs. 67 and 116.)

TEST NO. 45. STARRED ANGLE. 2-in. 1:2:4 MERAMEC RIVER GRAVEL CONCRETE, ROUND SECTION.—20 to 30 mins.—a number of deep cracks, mostly vertical, in lower half of column; concrete becoming loose in places. 35 mins.—some spalling near middle of column. 36 mins.—cracks in upper half of column; practically all sides have large cracks. 38 mins.—crack on west, 3 ft. up, open 3 in. exposing steel. 44 mins.—spalling to steel on west. 47 mins.—spalling to steel on north and west near bottom. 55 mins.—wire tie appeared to be holding concrete between angles in place. 56 mins.—edge of steel exposed on west from 1 ft. to 6 ft. up, on north from 1 to 2 ft. up. 1 hr. 7 mins.—steel exposed on south, 1 to 2 ft. up. 1 hr. 10 mins.—large pieces spalled on north side near middle of column. 1 hr. 34 mins.—concrete on lower part of bracket spalled off. 1 hr. 41 mins.—maximum expansion. 1 hr. 47 mins.—failure by local buckling about 3 ft. above base.

After failure.—Coarse aggregate was quite generally broken on fracture planes. Concrete inside of fractures appeared to be in good condition. (Figs. 66 and 112.)

TEST NO. 46. LATTICED ANGLE. 2-in. 1:2:4 TRAP CONCRETE.—1 hr.—diagonal crack, southeast corner at bracket. 2 hrs. 35 mins.—vertical crack, 3 in. long, on east face, 4 in. from southeast corner, 8 ft. up. No spalling before failure. 4 hrs. 55 mins.—maximum expansion. 6 hrs. 43½ mins.—failure with local buckling about 6 ft. above base, each angle deflecting outward about 2½ in.

After failure.—Incipient fusion on surface of concrete with small cracks. Concrete crumbly from dehydration to 1½ in. from surface. It was otherwise apparently in fair condition, except where crushed at point of failure. (Figs. 67 and 117.)

TEST NO. 47. ROUND CAST IRON. 2-in. 1:2:5 CINDER CONCRETE. NOT TIED.—50 mins.—piece of concrete 2 in. diameter, spalled off under bracket on south, exposing bracket; also crack 6 in. long extended down from spall. 2 hrs. 3 mins.—maximum expansion. 2 hrs. 42 mins.—vertical cracks noted on all sides at middle of column. 2 hrs. 47 mins.—concrete fell off from 2 to 10 ft. up, exposing metal. 2 hrs.—slight deflection to south noted, increasing to ¼ in. at 2 hrs. 20 mins. 2 hrs. 48¾ mins.—failure with buckling to south, maximum 6 ft. above base.

After failure.—Column broken up and thickness found to vary from ½ to $1\frac{3}{16}$ in., with thinnest metal on north. A short horizontal crack had formed on north side 3 ft. above base. Concrete crumbly for ⅜ in. from surface. Inside of this it appears almost unaffected by the fire test. (Figs. 67, 101, and 158.)

(*e*) HOLLOW CLAY-TILE PROTECTIONS

TEST NO. 48. ROLLED H. 2-IN. SEMIFIRE-CLAY TILE, NEW JERSEY DISTRICT. NO FILLING.—7 mins.—a number of vertical joints open about $\frac{1}{16}$ in. 12 mins.—vertical cracks in center of north, east, and south sides, 5 to 9 ft. up; cracks in about ⅓ of all vertical joints. 20 mins.—vertical cracks extending and in some cases opening to ¼ in.; cracks on all sides at bracket, and lower edge of tile spalled off on southwest corner at bracket. 30 mins.—north face cracked, 2 to 10 ft. up; vertical cracks on east from 3 to 10½ ft., on south from 5 to 9 ft. up; cracks in some cases open ⅜ in. 1 hr. 15 mins.—outer shell of tile spalled off along lower west edge at bracket; little change in tile below; no spalling below bracket before failure. 1 hr. 32 mins.—maximum expansion. 1 hr. 40 mins.—slight deflection to northwest noted, increasing to ⅝ in. at 1 hr. 49 mins. 1 hr. 50 mins.—failure, buckling to west, maximum 8 ft. above base.

After failing.—At failure tile fell off from 5 ft. up to top course of bracket. Lower two or three courses almost intact. (Figs. 68, 118, and 162.)

TEST NO. 49. ROLLED H. 4-IN. SEMIFIRE-CLAY TILE, NEW JERSEY DISTRICT. NO FILLING.—3½ to 8½ mins.—a number of vertical cracks in tile and joints in lower 8 ft. opened $\frac{1}{16}$ to $\frac{3}{16}$ in. 12 to 24 mins.—cracks opening ¼ to ⅝ in. maximum; west face bulged out ¾ in. from 4 to 8 ft. up. 22 mins.—ties down at 9th and 12th courses. 25 mins.—tile in middle section of column on south and west sides cracked and crushed. 1 hr. 27 mins.—cracks opening up to ⅞ in. in places; no extension of cracks or spalling. 1 hr. 20 mins.—slight deflection to north noted, increasing to ¼ in. at 1 hr. 38 min. 1 hr. 21 mins.—maximum expansion. 1 hr. 40 mins.—failure with compound buckling to northwest 8 ft. above base and to southeast 6 ft. above base.

After failure.—A few tile intact, without cracks; tile generally cracked longitudinally through the shells, not often cracked transversely across webs. This probably accounts for the fact that no spalling of shells occurred until very near failure. (Figs. 68, 118, and 162.)

TEST NO. 50. PLATE AND ANGLE. 2-IN. SURFACE-CLAY TILE, BOSTON DISTRICT. GRANITE CONCRETE FILL. (TILE, 6 IN. WIDE, LAID HORIZONTALLY.)—7 mins.—cracking and spalling on corners, 4 to 6 ft. up. 10 mins.—tile spalled, concrete exposed on south and east for 1 ft. near bottom. 16 mins.—tile bulged out on south face from bottom to 6½ ft. up. 21 mins.—tile fell off on south where bulged, exposing steel. 26 mins.—concrete exposed from 4 to 6½ ft. up on north, east, and west. 32 mins.—cracks opening up at joints in upper half. 55 mins.—maximum expansion. 1 hr. 5¾ mins.—failure, with local buckling about 5½ ft. above base.

After failure.—Tile broke clear from concrete with no splitting of tile at keys. Transverse shearing of long tile at joints of short tile very marked. Tile in all courses fell in whole or part except top bracket course and three lower courses. Concrete in fill where not crushed by compression apparently uninjured. (Figs. 68 and 119.)

TEST NO. 50A. PLATE AND ANGLE. 2-IN. SURFACE-CLAY TILE, BOSTON DISTRICT. GRANITE CONCRETE FILL. (TILE 6 IN. WIDE, LAID HORIZONTALLY.)—5 mins.—several horizontal joints open ⅛ in.; vertical crack 12 in. long through tile on east face, 6¾ ft. up; large number of vertical cracks near corners at bracket. 7 mins.—part of one outer shell on each of south and west, near middle spalled. 13 mins.—vertical cracks ¼ in. wide opposite joints at all corners. 13 to 20 mins.—outer shell spalled off for 2 ft. at middle on east and west; outer shell bulged out ½ in. in several places. 21 mins.—edge of steel exposed for 6 in. on northeast corner, 6 ft. up. 21 to 39 mins.—tile fell off, exposing concrete for 3 ft. at middle on east and west; buckling of tile increasing on flanges to ¾ in. maximum; upper 3 ft. intact, except for ⅛-in. cracks at bracket. 1 hr. 14 mins.—tile fell off, exposing concrete in lower 3 ft. on west face; otherwise little change. 1 hr. 24 mins.—mortar joints evidently not very full on north and south, can see through in places. 1 hr. 42 mins.—tile buckled out 1 in. on north, 5 ft. up. 1 hr. 44 mins.—maximum expansion. 1 hr. 59½ mins.—failure, buckling to west, maximum 5¼ ft. above base.

After failure.—All tile fell off at failure except parts of courses in lower foot and in upper 3 feet. Concrete fully fills reentrant portions except where crushed out at point of failure. (Figs. 69, 119, and 163.)

TEST NO. 51. PLATE AND ANGLE. 4-IN. SURFACE-CLAY TILE, BOSTON DISTRICT. GRANITE CONCRETE FILL.—7 to 17 mins.—considerable cracking and bulging, mostly in middle courses, where some corners were shattered. 14 mins.—vertical cracks in about 50 per cent of tile. 18 to 38 mins.—bulging increasing, vertical cracks opening to ¾ in. maximum; minor spalling on southeast and southwest corners in middle courses. 20 mins.—ties on 2d and 8th courses down. 36 mins.—tie on 6th course down. 40 to 52 mins.—outer shell of tile spalled off from 5 to 7 ft. up on south and

west. 1 hr. 22 mins.—bulging of tile increased to 1½ in. maximum. 1 hr. 47 mins.—tile fell on north, to 7 ft. up, exposing mortar and concrete. 1 hr. 30 mins.—slight deflection to west noted, increasing to ⅛ in. at 2 hr. 2 hrs. 11 mins.—maximum expansion. 2 hrs. 17¼ mins.—failure, buckling to west, maximum 6 ft. above base.

After failure.—Tile generally shattered. Parts of tile near bottom blackened and partly fused by the heat. Concrete fills web spaces fairly; it is apparently little injured. Mortar joint on flanges not full in places, but voids are partly filled with concrete. (Figs. 69 and 120.)

TEST NO. 51A. PLATE AND ANGLE. 4-IN. SURFACE-CLAY TILE, BOSTON DISTRICT. GRANITE CONCRETE FILL.—3 to 15 mins.—considerable cracking and some bulging at corners. 16 mins.—cracks quite general, open to ¼ in. in lower two-thirds, not over $\frac{1}{16}$ in. above. 16 to 44 mins.—cracks increasing and opening up; bulging out of tile at horizontal joints to maximum of 2 in., mostly in middle four courses. 38 mins.—tie broken on 5th course. 1 hr. 6 mins.—cracks and bulges opening up in lower 8 ft.; little change above. 1 hr. 14 mins.—ties dropped on 2d and 3d courses. 1 hr. 20 mins. to 1 hr. 32 mins.—outer shells on east and west sides in middle courses spalling; at end of this period tile had fallen, exposing concrete from 4 to 8 ft. up on east and from 4 to 10 ft. up on west. 1 hr. 44 mins.—all ties down except on three upper courses. 1 hr. 45 mins.—tile buckled away from flanges on north and south 6 ft. up. 2 hrs. 6 mins.—tile spalled on north 6 ft. up, exposing east flange 3 by 4 in. 2 hrs. 23 mins.—tile buckled away from flanges on north and south 8½ ft. up. 2 hrs. 26 mins.—no change in upper courses. 2 hrs. 40 mins.—maximum expansion. 2 hrs. 54 mins.—tile fell on south 6 ft. up, exposing steel. 2 hrs. 41 mins.—lateral deflection at center less than ⅛ in. 2 hrs. 55½ mins.—failure with buckling to west, maximum at 5½ ft. above base.

After failure.—Upper 2½ courses almost intact except for spalling of part of outer shell on west. Tile generally cracked through shells parallel with webs, but some were also cracked through webs parallel with shells. Filling good, no voids. Tile fused near bottom on south and west sides, some pieces had been nearly plastic. Partial fusion extended up to the 10th course. (Figs. 69, 120, and 163.)

TEST NO. 52. PLATE AND CHANNEL. 2-IN. OHIO SHALE TILE. CINDER-CONCRETE FILL.—2 mins.—cracks ⅛ in. wide on east and west. 8 to 11 mins.—cracks increasing and open to ¼ in., some shells buckling, two outer shells spalled, and considerable number of outer shells loose. 16 mins.—cracks open to 1 in. maximum; tie broken on 7th course. 18 mins.—outer shell 7th course on west spalled off. 27 mins.—tie broken on 9th course. 32 mins.—outer shell 9th course on south spalled; a number of loose outer shells held in place by ties. 53 mins.—all tile cracked, but little further spalling. 1 hr. 6 mins.—edge of southeast flange exposed opposite 6th course. 1 hr. 25 mins.—inner shell ready to fall on 7th course, west, edges of flanges probably exposed. 1 hr. 32 mins.—maximum expansion. 1 hr. 40¾ mins.—failure with local buckling about 6 ft. above base.

After failure.—Concrete fill without voids; mortar joints on north and south sides fairly full. Bracket courses almost intact. (Figs. 70, 121, and 163.)

TEST NO. 53. PLATE AND CHANNEL. 4-IN. OHIO SHALE TILE. CINDER-CONCRETE FILL.—2 to 15 mins.—cracking very pronounced on all faces, cracks opening up to 1 in. maximum at end of this period. 7 to 30 mins.—spalling of outer shells beginning at corners and later across faces; outer shells generally loose. 8 to 23 mins.—bulging out of tile at horizontal joints on east and west, 3 to 9 ft. up, increasing to 1½ in. at end of period. 24 mins.—ties off on 3d, 7th, and 9th courses. 30 to 33 mins.—all outer shells spalled off on west, 2 to 6 ft. up on east, 9 to 10 ft. up. 33 to 37 mins.—tile fell on west, 4th and 5th courses, on east 7th, 8th, and 9th courses. 42 mins.—all ties down except on 1st, 8th, 11th, and 12th courses. 45 mins.—tile bulged out on south, exposing steel from 2 to 5 ft. up. 58½ mins.—tile now fallen on west from 3d to 9th

courses, inclusive. 1 hr. 4 mins.—bracket courses nearly intact. 1 hr. 17 mins.—maximum expansion. 1 hr. 22¼ mins.—failure with local buckling about 7½ ft. above base.

After failure.—Cracks through tile webs parallel with faces predominate, although many units cracked through both faces. Mortar joints generally full. Concrete fill good, no voids. (Figs. 70, 121, and 163.)

TEST NO. 54. LATTICED CHANNEL. 2-IN. OHIO SEMIFIRE-CLAY TILE. TRAP CONCRETE FILL.—2 to 15 mins.—considerable cracking, both in joints and in tile, cracks opening up generally to ⅛ in. at end of period; outer shells spalling near corners and beginning to bulge out at horizontal joints, mostly in middle courses. 24 mins.—bulging out of outer shells increased to maximum of 2 in. 28 mins.—tie broken on 9th course. 48 mins.—nearly all of outer shells spalled off at bracket on south. 50 mins. to 1 hr. 4 mins.—outer shell of tile spalled on south, 4 to 5 ft. up, north half of east face, 3 to 5 ft. up, and on west 7 to 9 ft. up. 1 hr. 22 mins.—tie broken on 4th course. 1 hr. 41½ mins.—inner shells on east fell 4 to 6 ft. up, exposing steel. 1 hr. 43 mins.—inner shells on north fell 4 to 6 ft. up, exposing steel; outer shell spalled on north 3 to 4 ft. up. 1 hr. 52 mins.—tile fell on west 2 to 4 ft. up, exposing flange edges; on north, 3 to 6 ft. up, exposing part of flanges and lattice. 2 hrs. 30 mins.—all tile off on west, 9th and 10th courses. 2 hrs.—slight defection to northwest noted, increasing to ⅛ in. at 2 hrs. 41 mins. and to 1 in. west at 3 hrs. 1 min. 2 hrs. 20 mins.—maximum expansion. 3 hrs. 17¼ mins.—failure, buckling to west, maximum 6 ft. above base.

After failure.—Concrete between flanges of channels quite crumbly; that between channel webs hard and apparently little injured except where crushed. Concrete fairly fills space between steel and tile on both flange and web sides. (Figs. 70, 122, and 164.)

TEST NO. 55. Z BAR AND PLATE. 2-IN. OHIO SEMIFIRE-CLAY TILE. LIMESTONE CONCRETE FILL.—4 mins.—cracking and bulging out of outer shells. 13 to 23 mins.—outer shells generally cracked to ⅜ in. maximum; bulging out at horizontal joints increasing; some spalling at corners. 34 mins.—parts of shell on west spalled at 4 to 5 ft. and 2 to 3 ft. up. 45 mins.—all cracks opening. 1 hr. 4 mins.—wire tie broken on 3d course. 1 hr. 48 mins.—tile fell on west, 7th to 10th courses, inclusive, also part south. 2 hrs.—outer shell spalled on south at bracket course exposing edge of bracket steel. 2 hrs. 8 mins.—outer shell spalled on north, 7th to 9th course. 2 hrs. 37 mins.—tile bulged out 1¼ in. on south 9 ft. up. 2 hrs. 51 mins.—maximum expansion. 3 hrs. 20 mins.—tile fell on west 11th course. 3 hrs. 23 mins. to 3 hrs. 27 mins.—outer shells spalled on north at bracket and from 2 to 6 ft. up; on east, shells now spalled from 2 to 7 ft. up. 3 hrs. 41 mins.—deflection of ⅛ in. south. 3 hrs. 46¾ mins.—failure, buckling to south, maximum at 8 ft. above base.

After failure.—Tile shells remaining on are firmly held. Concrete filled interior fully, including space between Z bar flanges and tile. Concrete hard where tile remained in place; where exposed concrete was calcined to depth of about ½ in. (Figs. 71, 122, and 164.)

TEST NO. 56. Z BAR AND PLATE. 4-IN. OHIO SEMIFIRE-CLAY TILE. LIMESTONE-CONCRETE FILL.—5 to 30 mins.—considerable cracking and some spalling of parts of outer shells; spalling principally in upper half near southwest corner, at middle near northwest corner, and on lower courses at northeast corner. 57 mins.—outer shell bulged slightly at some horizontal joints, most on west at 6 and 7 ft. up; no inner shells spalled. 2 hrs. 30 mins.—maximum expansion. 2 hrs. 55 mins.—outer shell spalled on south at bracket; otherwise little cracking or spalling in last 2½ hrs.; wire mesh in horizontal joints holding tile in place quite effectively. 1 hr. 51 mins.—

deflection of ⅛ in. south; deflection changed to ⅛ in. northeast at 3 hrs. 1 min., increasing to ⅝ in. at 3 hrs. 21 mins. 3 hrs. 33½ mins.—failure, buckling to north, maximum at 8½ ft. above base.

After failure.—Wire mesh in all horizontal joints lapping at corners. Concrete fill good, no voids; concrete hard, almost intact. (Figs. 71, 123, and 165.)

TEST NO. 57. I BEAM AND CHANNEL. 4-IN. SURFACE-CLAY TILE, CHICAGO DISTRICT. LIMESTONE-CONCRETE FILL.—3 to 13 mins.—general cracking and bulging of a few outer shells, cracks open to ⅛ in. 23 mins.—bulging increasing; general spalling of small pieces from corners. 25 mins.—parts of outer shell spalled on east and west 4 to 5 ft. up. 32 mins.—cracks and bulges generally open ½ in. on all sides from 3 to 6 ft. up. 38 mins.—outer shells loose or fallen on west 5 to 7 ft. up and on south 4 to 6 ft. up. 53 mins.—all outer shells spalled on east and south 3 to 6 ft. up, and on north 5 to 6 ft. up; tile in 5th course on south fell, exposing part of flange. 1 hr. 2 mins.—large amount of tile fell on all sides, exposing most of concrete from 3 to 10 ft. up. 2 hrs. 15 mins.—maximum expansion. 2 hrs. 37 mins.—tile down on all sides from 2 to 12 ft. up, exposing concrete. 2 hrs.—slight deflection to north noted, increasing to ¼ in. at 2 hrs. 50 mins., and to ½ in. at 3 hrs. 21 mins. 3 hrs. 23 mins.—failure, buckling to north, maximum at 8 ft. above base.

After failure.—A number of tile sheared at plaster key, leaving latter in the concrete. Concrete fill fairly full, except at a few points between tile and channel flange. Concrete calcined to depth of ½ in. where exposed; hard where protected. (Figs. 71 and 123).

TEST NO. 58. I BEAM AND CHANNEL. 2 LAYERS OF 2-IN. SURFACE-CLAY TILE, CHICAGO DISTRICT. TILE FILL. WIRE MESH IN HORIZONTAL JOINTS.—3 to 15 mins.—outer shells and tile spalled at corners quite generally. 16 mins.—outer shell of outer tile bulged out 1½ in. at horizontal joint on east 5 ft. up. 26 mins.—part of inner shell of outer layer of tile at southeast corner 10 ft. up fell; outer shells continuing to spall at corners and on parts of some faces. 44 to 48 mins.—outer shells of outer tile spalled on all sides at bracket, also partly on east at 10 ft. up, and all on west 10 to 11 ft. up; outer shell bulged 1 in. at 10½ ft. up on south. 1 hr. 36 mins.—all outer shells now spalled on east from 9 ft. to top. 1 hr. 57 mins.—outer tile down on south, 10th course, and on east, 9th course. 3 hrs.—little change in 1 hr. 3 hrs. 30 mins.—outer shell of outer tile spalled on south, 5th course. 3 hrs. 43 mins.—bracket steel exposed for 6 in. on northeast. 3 hrs. 45 mins.—all tile down in upper two courses on east; on north and south, outer tile down in upper two courses and inner tile bulging out 1 in.; all of bracket exposed on both sides. 3 hrs. 56 mins.—50 to 75 per cent of outer shells of outer tile now down, most of which occurred before 1 hr. 57 mins.; total outer tile and part of inner tile fallen in a few places as noted above; little spalling now taking place. 4 hrs.—maximum expansion. 4 hrs. 25 mins.—no decided change; no fusion noted, although tile at 4 ft. up concave outward as if beginning to fuse. 3 hrs.—slight center deflection to south noted, increasing to ¼ in. at 4 hrs. 31 mins. 4 hrs. 35¾ mins.—failure, with local buckling to south and west about 11 ft. above base.

After failure.—Generally all but outer shell of outer tile in place in lower eight courses. On 9th course outer tile down and almost all tile down in 10th, 11th, and 12th courses. Wire mesh found in all joints except between the 1st and 2d courses, between the 9th and 10th on north and south where the pipes interfered, and between the 11th and 12th where the bracket angles interfered. Some mesh was found between the 10th and 11th courses, although so much tile was down that it was impossible to determine whether all pieces had been placed. Very decided fusion of tile occurred in lower 4 ft. and incipient fusion up to 7 ft. above base, being most pronounced on south side of column. (Figs. 71, 124, and 165.)

TEST NO. 59. I BEAM AND CHANNEL. 2 LAYERS OF 2-IN. SURFACE-CLAY TILE, CHICAGO DISTRICT. TILE FILL. OUTSIDE WIRE TIES.—3 to 14 mins.—considerable cracking of outer shells on all sides. 14 mins.—outer tile bulged out 1½ in. on west, 7 ft. up. 15 to 23 mins.—spalling of two outer shells at corners. 17 mins.—outer tile bulged out ½ in. on north, 4 ft. up, increasing to 2 in. at 27 mins. 26 mins.—nearly all of outer shells of outer tile spalled on west, 6 to 8 ft. up; tie broken on 8th course. 29 mins.—outer shell spalled on east, 7th and 8th courses. 33 mins.—all outer tile 8th and 9th courses down and part of tile in 7th course. 35 mins.—inner tile bulged out 2½ in. on west, 7 ft. up. 36 mins.—outer tile loose on west in 11th and 12th courses, held by wire. 38 mins.—all outer tile down on south from 3 to 6 ft. up, on north from 3 to 8 ft. up. 42 mins.—inner tile on south bulged out ¾ in., 5 ft. up. 1 hr. 6 mins.—inner tile down on south, 6 to 9 ft. up, exposing both flanges. 1 hr. 16 mins.—all outer tile and parts of inner tile down except for parts of upper and lower courses. 1 hr. 23 mins.—all tile down on west, 6 to 8 ft. up, exposing channel. 1 hr. 25 mins.—maximum expansion. 1 hr.—slight center deflection to north noted, increasing to ⅜ in. at 1 hr. 31 mins. 1 hr. 33¾ mins.—failure with buckling to north, maximum at 7½ ft. above base, and twisting about 22°.

After failure.—All but filling tile down in middle six courses. Filling tile also thrown off near middle of column. All tie wires broken before or at failure. (Figs. 72 and 124.)

TEST NO. 60. LATTICED ANGLE. 2-IN. OHIO SEMIFIRE-CLAY TILE. TRAP-CONCRETE FILL, PLACED BEFORE TILE WAS SET.—3 to 18 mins.—cracking of tile on all sides; verical crack on north near west corner open to ¾ in., 3 to 6 ft. up. 13 to 18 mins.—ties broke on 9th, 6th, 8th, 4th, and 3d courses, in the given order. 18½ mins.—part outer shells spalled on south 3 to 6 ft. up. 26 mins.—tile down on west from 1 to 7 ft. up, exposing concrete. 32 mins.—tile bulged out 2 in. on north 2 to 6 ft. up. 33 mins.—tile down on north from 1 to 6 ft. up. 36 mins.—tile bulged out 1½ in. on east, 2 to 7 ft. up. 38 mins.—a few cracks in tile in upper three courses. 40 mins.—all tile down on east 1 to 10½ ft. up. 48 mins.—tile down on north, 7¼ to 10½ ft. up. 1 hr. 2 mins.—tile down on south, 7th course. 1 hr. 47 mins.—maximum expansion. 1 hr. 58 mins.—little change during last hour. 2 hrs. 7 mins.—all tile down on south up to bracket. 2 hrs. 13 mins.—$\frac{1}{16}$-in. vertical cracks in concrete halfway up increasing to ⅛ in. at 2 hrs. 24 mins. 2 hrs. 21 mins.—center deflection of ⅛ in. northwest, increasing to ¾ in. at 3 hrs. 7 mins. 3 hrs. 9½ mins.—failure with local buckling 4½ to 5 ft. above base.

After failure.—Tile quite generally cracked vertically through both shells, transverse cracks along webs not general. Mortar evidently did not bond tile and concrete. Except at point of failure, cracks in concrete are not wider than $\frac{3}{16}$ in. Concrete very crumbly outside lattice, quite hard inside. (Figs. 72, 125, and 166.)

TEST NO. 61. LATTICED ANGLE. 2-IN. OHIO SEMIFIRE-CLAY TILE. NO FILLING.—3 to 18 mins.—pronounced cracking and bulging especially at middle courses; cracks mostly vertical, opening to 1¼ in. maximum on south, 5 to 7 ft. up; very little spalling. 13 mins.—ties broken on 4th, 6th, and 7th courses. 26 mins.—both outer and inner shells fell on northwest corner 4 ft. up for width of about 4 in. 32 mins.—similar spall on southeast corner 4 ft. up. 35 mins.—considerable bulging of tile on south, 3 to 4 ft. up. 44 mins.—cracks and bulging increased slightly, none now more than ¼ in. 45 mins.—maximum expansion. 47 mins.—center deflection of ¼ in. east. 50¼ mins.—failure with local buckling about 6 ft. above base.

After failure.—Tile cracked transversely in some cases; greater number cracked vertically through both faces without transverse web cracks. (Figs. 73 and 125.)

TEST NO. 62. ROUND CAST IRON. 2-IN. POROUS SEMIFIRE-CLAY TILE, NEW JERSEY DISTRICT; NO FILLING.—Air pressure tanks shut off, load applied by water pressure only. 6 to 20 mins.—a number of vertical cracks in tile and in mortar joints, opening to maximum of ¼ in. at end of this period. 45 mins.—cracks now in 40 to 50 per cent of tile units, width varying from fine to ⅜ in. in column proper and to ½ in. at bracket. 2 hrs.—all ties in place; cracks have opened up slightly. 2 hrs. 25 mins.—horizontal cracks on west between bracket courses about $\frac{3}{16}$ in. wide. 2 hrs. 42 mins.—similar cracks on southwest, ½ in. wide. 3 hrs.—maximum expansion. 3 hrs. 26 mins.—horizontal bracket cracks open to 1 in. maximum. 4 hrs. 10 mins.—cracks open 1 in. at middle of column; no spalling or bulging of tile. 4 hrs.—slight deflection to southeast noted, increasing to $1\frac{3}{8}$ in. at 4 hrs. 10 mins. 4 hrs. 11½ mins.—failure, column unable to support working load. 4 hrs. 13 mins.—load of about 75 000 lbs. held. 4 hrs. 14 mins.—tile fell on east; load of about 25 000 lbs. held. 4 hrs. $14\frac{3}{4}$ mins.—gas shut off; column buckled to southeast maximum at 8 ft. above base.

After failure.—Column did not crack although metal buckled near surface on compression side at failure point. Cracks in tile are generally vertical, extending through both shells. Tie wires were greatly oxidized and had little strength. (Figs. 73, 126, and 167.)

TEST NO. 63. ROUND CAST IRON. 2-IN. POROUS SEMIFIRE-CLAY TILE, NEW JERSEY DISTRICT; NO FILLING.—6 mins.—vertical crack on east, 4 to 5 ft. up, $\frac{1}{32}$ in. wide. 8 to 11½ mins.—gas shut off accidentally during this period. 13½ to 23½ mins.—fine vertical cracks 2 to 6 ft. long opening up to ¼ in. maximum. 32 mins.—outer shell loose on 3d course, east side. 35 mins.—cracks open to ½ in. 47½ mins.—cracks open to ⅞ in. maximum. 1 hr. 25 mins.—cracks opened but slightly during past 30 mins. 1 hr. 37 mins.—some flaking of outer shells; no spalling. 1 hr. 55 mins.—very little change; tile above 10 ft. up almost intact except for a few cracks ⅛ in. wide. 2 hrs. 20 mins.—maximum expansion. 2 hrs. 23 mins.—about ⅓ of outer shell spalled on 3d course, east side. 2 hrs. 24 mins.—bulging on southeast, 4 to 6 ft. up, with spalling of parts of both shells leaving crack ¾ to 2 in. wide. 2 hrs. 41 mins.—slight center deflection to north, increasing to 1¼ in. at 2 hrs. 55 mins. 2 hrs. 57½ mins.—failure with buckling to north, maximum at 5½ ft. above base.

After failure.—Most cracks in tile were vertical, straight through both shells. Tie wires greatly oxidized, leaving $\frac{1}{16}$ in. effective diameter. At failure point 5½ ft. up, cast iron was mushroomed out about 1 in. for one-half the circumference in a height of 2½ in. Numerous vertical cracks in the iron formed in this region. Tension breaks extend for one-half of the circumference on south, 3½ ft. above the base and 1¾ ft. below the head. The thickness of metal at the center break varied from ½ to 1¼ in., the thinnest metal being on the south. (Figs. 73, 126, and 167.)

TEST NO. 76. ROLLED H. 2-IN. HOLLOW CLAY TILE COVERED WITH ¾-IN. LAYER OF GYPSUM PLASTER; LIMESTONE CONCRETE FILL. UPPER 4 COURSES, OHIO SHALE; MIDDLE 4 COURSES, OHIO SEMIFIRE CLAY; LOWER 4 COURSES, SEMIFIRE CLAY, NEW JERSEY DISTRICT.—1 to 3 mins.—outer coat of plaster peeled off and shattered at end of this period; vertical cracks extending through to tile in places. 10½ mins.—inner coat of plaster fell on south, exposing tile 3 to 8 ft. up. 11 to 32 mins.—inner coat of plaster fell, exposing about two-thirds of tile at 21 mins.; at 32 mins. about 90 per cent of plaster had fallen below bracket course; plaster still on bracket on all four sides and down to 9 ft. up on west. 28 mins. to 1 hr. 10 mins.—some spalling of outer shells and very slight amount of cracking, mostly at corners; at end of this period northeast corner was spalled 5 to 6 ft. up, also on west 4 to 5 ft. up, and on east 6 to 7 ft. up; shell loose on north 7 to 8 ft. up. 1 hr. 32 mins.—several corners spalled in upper

two thirds of column. 1 hr. 41 mins.—outer shell spalled on north 8th course; plaster on at bracket, except on north side. 2 hrs. 30 mins.—little change during past 50 mins., except for a few spalls of outer shells in upper half. 3 hrs. 30 mins.—no change except outer shell spalled on west, 9th course. 3 hrs. 45 mins.—maximum expansion. 4 hrs.—little change, except for a few vertical cracks, $\frac{1}{16}$ to $\frac{1}{8}$ in., near the top; plaster still on parts of bracket. 4 hrs. 18 mins.—outer shell spalled on east, 9th course; no inner shells spalled before failure; no spalling of outer shells on four lower courses; deflections not measured. 4 hrs. 25½ mins.—failure with local buckling about 9½ ft. above base.

After failure.—Shale tile almost all off; could not be examined; Ohio semifire-clay tile quite shattered, no bond with mortar or fill; semifire clay, New Jersey district, tile in fairly good condition except for a number of fine cracks; fair bond with fill. Concrete fill full, no voids. Mortar joints on sides were fairly full. (Figs. 78, 138, and 46.)

TEST NO. 77. PLATE AND ANGLE. 4-IN. HOLLOW CLAY TILE COVERED WITH $\frac{5}{8}$ IN. LAYER OF LIME PLASTER; LIMESTONE-CONCRETE FILL. UPPER 4 COURSES, SEMIFIRE CLAY, NEW JERSEY DISTRICT; MIDDLE 4 COURSES, SURFACE CLAY, CHICAGO DISTRICT; LOWER 4 COURSES, SURFACE CLAY, BOSTON DISTRICT.—$\frac{1}{2}$ min.—most of plaster spalled off, exposing nearly three-fourths of total tile surface. 15 mins.—practically all of plaster down, except small amount in places near top; some outer shells of tile beginning to buckle in middle courses. 20 to 50 mins.—cracking and some spalling of outer shells, mostly at corners, in middle 4 courses; a few cracks in lower 4 courses, less than $\frac{1}{16}$ in. wide. 1 hr. 10 mins.—very little change during 20 mins., though cracks opened slightly; no cracks observed in upper 4 courses. 1 hr. 35 mins.—a few fine cracks in upper 4 courses, not over $\frac{1}{32}$ in. 1 hr. 49 mins.—a number of horizontal cracks in lower 4 courses up to $\frac{1}{8}$ in. wide; outer shell spalled on west 3 to 4 ft. up. 2 hrs. 20 mins.—horizontal and vertical cracks in lower 4 courses opening up $\frac{1}{8}$ to $\frac{3}{16}$ in.; cracks in upper 4 courses not opening up. 3 hrs. 22 mins.—a few tile in lower courses curved out, splitting near ends; not much change in middle 4 courses. 4 hrs.—cracks in lower 4 courses open $\frac{3}{16}$ to $\frac{1}{2}$ in.; northeast corner spalled 2 to 3 ft. up; possibly fusion 3 ft. up. 4 hrs. 10 mins.—maximum expansion. 4 hrs. 17 mins.—more spalling of outer shells in middle courses; no inner shells spalled. 4 hrs. 20 mins.—tile bulged out from steel just above middle on north and south sides; very decided fusion 1 to 3 ft. up. 4 hrs. 26 mins.—cracks in lower 4 courses open $\frac{5}{16}$ to $\frac{1}{2}$ in.; outer shell 5th to 7th course, north side, spalled; cracks in upper 4 courses open not over $\frac{1}{32}$ in. 4 hrs. 26 mins. to 4 hrs. 30 mins.—considerable spalling of outer shells in middle 4 courses; all inner shells, all courses, in place before failure. Deflections not measured. 4 hrs. 42¼ mins.—failure with buckling to west, maximum at 6 ft. above base.

After failure.—All of upper four courses and middle four courses of tile down, examination not possible. Lower four courses much shattered by vertical and horizontal cracks; incipient fusion through outer shells and through webs where exposed; pieces of this tile bent, curled, and discolored. Concrete fills reentrant portions fully. Appearance of steel indicated that concrete had not fully filled space between flanges and tile. (Figs. 78, 139, and 46.)

(*f*) GYPSUM BLOCK PROTECTIONS

TEST NO. 64. ROLLED H. 4-IN. WESTERN GYPSUM BLOCK (SOLID). HOLLOW GYPSUM-BLOCK FILL.—10 mins.—fine surface checks developing, about $\frac{1}{32}$ in. wide. 30 mins.—fine surface checks all over faces, about ½ in. apart both ways. 48 mins.—corners spalled not over ½ in. by ½ in. 1 hr.—very little change. 1 hr. 30 mins.—surface checks about $\frac{1}{8}$ in. wide by $\frac{1}{8}$ in. deep; a few vertical joints opened not over

$\frac{1}{32}$ in. 2 hrs. 30 mins.—both horizontal and vertical joints opened about $\frac{1}{8}$ in.; mortar joints project $\frac{1}{4}$ in. 2 hrs. 40 mins.—block fell from bracket on north, exposing west part of bracket steel. 3 hrs. 10 mins. to 3 hrs. 25 mins.—spalling on southwest and northeast corners, just below middle, to 2 in. back. 3 hrs. 35 mins.—surface checks $\frac{1}{8}$ in. wide by $\frac{1}{2}$ in. deep; all cracks open to about $\frac{1}{4}$ in. 3 hrs. 45 mins.—mortar joints throughout project about $\frac{1}{2}$ in. 3 hrs. 55 mins.—piece of block down on southwest side of bracket; blocks on northeast and southeast sides of brackets stand out. 4 hrs. 5 mins.—block down on northeast side of bracket, exposing rest of bracket steel. 4 hrs. 10 mins.—mortar joints project $\frac{3}{4}$ in.; tile appears to be about 3 in. thick. 4 hrs. 15 mins.—block down on southeast side of bracket; both sides of south bracket now exposed; bracket steel glowing dull red. 4 hrs. 20 mins.—blocks on east and west faces tilting out from steel at 7 ft. up; surface checks appear to be about 1 in. deep. 4 hrs. 30 mins.—nearly all of block down on west 6 to 7 ft. up; all fell on east, from 4 ft. up to top of column, on south from 1 to 6 ft. up, exposing flange for 5 ft., also down on north, 10 to 11 ft. up. 4 hrs. 32 mins.—maximum expansion. 4 hrs. 39 mins.—all blocks down on west from 1 ft. up to top. 4 hrs. 30 mins.—slight deflection to northeast noted, increasing to $\frac{3}{8}$ in. at 4 hrs. 41 mins. 4 hrs. 43$\frac{1}{4}$ mins.—failure with buckling to east, maximum at 5 ft. above base.

After failure.—Exposed column flange on south greatly oxidized during test. Gypsum blocks shrunken to thickness of 3$\frac{1}{4}$ to 3$\frac{1}{2}$ in. Surface of blocks checked into $\frac{1}{4}$-in. squares, cracks extending inward to depth of 1$\frac{1}{4}$ to 2 in. from surface on sides and to 3$\frac{1}{2}$ at corners, the corner cracks extending in diagonally. Gypsum in this zone is brownish green. Further in gypsum is much more crumbly than new material, showing that the heat had affected it. (Figs. 74 and 127.)

TEST NO. 65. PLATE AND CHANNEL. 2-IN. WESTERN GYPSUM BLOCK (SOLID); SOLID GYPSUM-BLOCK FILL.—11 mins.—fine surface checks beginning to develop. 23 mins.—a few fine cracks and slight corner spalls; surface checks more distinct, forming in $\frac{1}{2}$ in. squares. 29 mins.—a few vertical joints open $\frac{1}{16}$ in. 48 mins.—surface checks $\frac{1}{16}$ in. wide by $\frac{1}{4}$ in. deep, distinct throughout courses; cracks $\frac{1}{2}$ in. deep; some corners spalled $\frac{1}{2}$ in. back, corners generally ragged. 1 hr. 5 mins.—cracks in all joints about $\frac{1}{16}$ in. wide; surface checks $\frac{1}{8}$ in. wide. 1 hr. 30 mins.—shrinking of tile apparent, mortar joints project $\frac{1}{16}$ to $\frac{1}{8}$ in.; cracks in joints now $\frac{1}{8}$ in. wide. 1 hr. 45 mins.—block fell on west 4 to 5 ft. up, exposing edges of flanges. 1 hr. 49 mins.—block fell on south 5 to 6 ft. up; mortar in joints projects out generally, $\frac{1}{4}$ to $\frac{3}{8}$ in. 2 hrs.—blocks fell on north 2 by 6 ft. up, steel still covered by mortar. 2 hrs. 5 mins.—blocks fell on east 2 to 6 ft. up, on west 2 to 5 ft. up; three top courses still intact. 2 hrs. 9 mins.—blocks down on south and west 5 to 6 ft. up. 2 hrs. 10 mins.—edges of flanges exposed where blocks have fallen, but steel plates are covered with about 1 in. of mortar in all cases. 2 hrs. 12 mins.—block down on west 6 to 7 feet up; piece fallen on northwest corner at bracket. 2 hrs. 17 mins.—mortar in joints projects $\frac{3}{8}$ in. 2 hrs. 20 mins.—maximum expansion; mortar fallen on north exposing steel; blocks down on north and west 7 to 8 ft. up. 2 hrs. 11 mins.—center deflection less than $\frac{1}{8}$ in. 2 hrs. 21$\frac{1}{2}$ mins.—failure with local buckling about 3 ft. above base.

After failure.—Blocks shrunken to thickness of 1$\frac{1}{2}$ to 1$\frac{3}{4}$ in. Surfaces of blocks checked into $\frac{1}{4}$ to $\frac{1}{2}$ in. squares, cracks being $\frac{1}{16}$ to $\frac{1}{8}$ in. wide at surface and $\frac{1}{2}$ to $\frac{3}{4}$ in. deep on sides; cracks at corners extending in diagonally 1 to 1$\frac{1}{4}$ in. Outer dehydrated surface quite firm and greenish in color in places; hardness extends to inner zone of checking where color changes from white to brownish. Gypsum farther in is softer than material not exposed to fire. Galvanizing on corrugated sheet-metal ties placed in the joints was removed; otherwise they were not oxidized or corroded. (Figs. 74 and 128.)

TEST NO. 66. LATTICED CHANNEL. 2-IN. EASTERN GYPSUM BLOCK (SOLID); POURED GYPSUM FILLING.—15 mins.—fine surface checks beginning to develop. 30 to 40 mins.—surface checks more distinct, forming in ¼ to 1 in. squares, mostly in lower half. 1 hr. 15 mins.—joints open to 1/16 in. increasing to ⅛ in. at 1 hr. 35 mins. 1 hr. 35 mins.—mortar in joints project 1/16 in. 2 hrs.—surface checks now cover whole surface, and are ¼ in. to ½ in. in depth; joints open to ⅜ in. maximum. 2 hrs. 5 mins.—block on east 5½ ft. up, tilting out ¾ in. at top. 2 hrs. 14 mins.—small spall on southwest corner at bracket, block below bracket on west tilting out at top. 2 hrs. 15 mins.—block down on east in 4th course, exposing lattice bars where unprotected by filling. 2 hrs. 22 mins.—block fell on east 8th course, exposing steel. 2 hrs. 26 mins.—blocks fell on north 1½ to 5½ ft. up, exposing steel. 2 hrs. 30 mins.—block fell on south, 4th course up, exposing steel. 2 hrs. 32 mins.—maximum expansion; blocks fell on south and west 2d course up. 2 hrs. 33 mins.—center deflection less than ⅛ in. 2 hrs. 36 mins.—failure with local buckling about 3½ ft. above base, causing column to deflect to west.

After failure.—Filling fairly full, but has some pockets, 2 in. deep; filling extends out between lattice bars, but not much beyond; very crumbly where exposed. Blocks shrunken to thickness of 1⅞ ins.; little change in color. Surface checks are 3/16 in. wide at surface and ½ in. to ¾ in. deep. Strips of wire mesh in joints not greatly oxidized. (Figs. 74 and 128.)

TEST NO. 67. ROLLED H. 4-IN. EASTERN GYPSUM BLOCK (SOLID); POURED GYPSUM FILLING.—17 mins.—fine surface checks, forming in 1-in. squares beginning to show. 34 mins.—surface checks more marked, 1/32 to 1/16 in. wide; checking most pronounced in lower half. 1 hr. 5 mins.—surface checks ¼ in. deep and ⅛ to ¼ in. wide; corner cracks ½ to 1 in. deep and 1 to 2 in. long. 1 hr. 20 mins.—joints beginning to open up; several horizontal cracks 1 in. deep across faces; surface checks now forming in ½ to ¾ in. squares. 1 hr. 50 mins.—very little change; surface checks ¼ to ½ in. deep are general. 2 hrs. 10 mins.—a few mortar joints project 1/16 to ⅛ in. 2 hrs. 21 mins.—pieces of corners at bracket spalled. 2 hrs. 40 mins.—joints open to ⅛ in.; surface checks ¾ in. deep and ¼ in. wide. 3 hrs. 45 mins.—mortar in joints projects ¼ in., vertical joints open to ¼ in., little change in surface checks. 4 hrs. 40 mins.—mortar projects to ½ in.; otherwise little change. 4 hrs. 52 mins.—top of block tilts out 1 in. on north 8½ ft. up. 4 hrs. 54 mins.—blocks fell on south 3d and 4th courses; mortar still covers steel quite generally; piece of block 4 in. wide also down at southwest corner in 5th course; joints near top open 3/16 in. 5 hrs. 1 min.—maximum expansion. 5 hrs. 5 mins.—blocks next to those fallen tilt away from steel; steel now exposed on south 3 to 4 ft. up. 5 hrs. 7 mins.—blocks down on west 3d to 6th course, exposing filling and flange edges. 5 hrs. 13 mins.—block fell on west 2d course; west edge of flange exposed 4 to 6 ft. up; little change after this until failure. 5 hrs. 31½ mins.—failure with local buckling about 4 ft. above base.

After failure.—Fill fairly full, but has some pockets 2 in. deep; mortar where still in place, covers flanges ¾ to 1 in.; steel fluxed for 30 in.-length near point of failure; edges of flanges attacked where exposed, unaffected where not exposed; strips of wire mesh oxidized through where they had been exposed to the fire; blocks shrunk to thickness of 3⅝ to 3⅞ ins. Surface check cracks formed in ¼ to ½ in .squares, cracks 1/16 to 3/16 in. wide at surface and generally 2 in. deep, although some extend completely through blocks; radial cracks at corners 2¼ in. deep. Block colored brownish yellow in outer 1 in., further in stained dark from oxidation of wood fiber. (Figs. 75 and 129.)

TEST NO. 67 A. ROLLED H. 4-IN. EASTERN GYPSUM BLOCK (SOLID); POURED GYPSUM FILLING.—15 mins.—fine surface checks beginning to show; fine cracks on ends of blocks. 45 mins.—surface checks forming in ¼ to 1 in. squares, most pronounced on west. 1 hr. 30 mins.—vertical joints opening slightly. 2 hrs.—cracks in both vertical

and horizontal joints opening up perceptibly; surface checks ¼ in. deep. 3 hrs.—vertical joints near middle of column open to ¼ in. 3 hrs. 30 mins.—all joints open from ⅛ to ¼ in.; mortar joints project ¼ in.; minor spalling from corners. 4 hrs.—all blocks have shrunk considerably; joints open ⅛ to ¼ in. 4 hrs. 30 mins.—small pieces spalling from corners; larger piece fallen from northwest corner, 7 ft. up. 5 hrs. 15 mins.—edge of bracket steel exposed on south side. 5 hrs. 15 mins. to 5 hrs. 45 mins.—considerable spalling at corners. 5 hrs. 45 mins.—maximum expansion. 5 hrs. 56 mins.—blocks down on north 2d and 3d courses, and on west 2d course; steel exposed. 6 hrs. 14 mins.—blocks now down on north from bottom to 5½ ft. up. 6 hrs. 20 mins.—blocks down on east 2d course and on south 3d course. 6 hrs. 21 mins.—block down on west 7th course. 6 hrs. 10 mins.—slight deflection to southwest increasing to ¼ in. at 6 hrs. 16 mins. 6 hrs. 24½ mins.—failure with buckling to south, maximum at 4¼ ft. above base.

After failure.—Filling fairly full. Column flange on north greatly oxidized for 3 ft. near point of failure. Blocks shrunk to thickness of 3½ to 3⅝ in.; gypsum very crumbly. Surface checks ¼ in. wide, extend in to depth of 2½ in. (Figs. 75 and 130.)

(*g*) BRICK PROTECTIONS

TEST NO. 68. ROLLED H. 2¼-IN. CHICAGO COMMON BRICK LAID ON EDGE; BRICK FILL.—13 mins.—a number of $\frac{1}{16}$-in. cracks in joints. 15 mins.—some cracking and spalling at corners. 16 to 18 mins.—several vertical cracks developing in middle sections, one on north face ⅜ in. wide, 2 ft. long. 23 mins.—brick fell on east from 3½ to 8½ ft. up, exposing north flange to depths of from 1 to 3 in. back from edge. 26 mins.—brick fell on west, 4 to 8½ ft. up, exposing south flange for width of 1 in., bricks bulged out ¼ in. from flanges on north and south in this region, increasing to ½ in. at 36 mins. 36 mins.—brick in upper third cracked somewhat; brick almost intact up to 3½ ft. above base. 1 hr. 4 mins.—very little change, bulging in middle slightly increased. 1 hr. 10 mins.—maximum expansion. 1 hr. 20 mins.—slight deflection to southeast noted, increasing to ¼ in. at 1 hr. 37 mins. 1 hr. 40¾ mins.—failure with local buckling 6½ to 8 ft. above base.

After failure.—Brick soft and crumbly and cracks readily. Mortar joints apparently quite full. (Figs. 76, 118, and 166.)

TEST NO. 69. ROLLED H. 3¾-IN. CHICAGO COMMON BRICK LAID FLAT. BRICK FILL.—*First Test.*—Trouble with one gas burner developed soon after start of test and gas was shut off at 37¼ mins. Test postponed for two days. Column observed after test was discontinued. One-half to two-thirds of brick cracked through vertically in one or more places, cracks from very fine to $\frac{1}{16}$ in. wide. Slight flaking and spalling of corners noted. Temperature of steel at end of test 45° C, attaining a maximum of 140° C, 3 hrs. 20 mins. later.

Second Test.—1 hr. 30 mins.—slight flaking, no spalling; cracks developed in first test not opening up. 3 hrs. 30 mins.—no cracks over $\frac{1}{16}$ in., brick flaking off at corners. 3 hrs. 47 mins.—cracks are vertical and generally very fine, maximum $\frac{1}{16}$ in. and not more than 2 ft. in length. 4 hrs. 16 mins.—very little change, cracks open to ⅛ in. maximum; no fusion. 4 hrs. 52 mins.—decided fusion in lower 3 ft. only. 5 hrs. 20 mins.—maximum expansion. 5 hrs. 38 mins.—fusion extended up to 6 ft., fused brick run down to base at corners. 6 hrs. 10 mins.—fusion up to 10 ft. above base. 6 hrs. 27 mins.—fusion extends up to 11 ft. above base; no spalling, except surface flaking, occurred before failure. 6 hrs.—slight center deflection to west noted, increasing to ⅝ in. at 7 hrs. 11 min. 7 hrs. 13¼ mins.—failure, with buckling to west, maximum at 5¼ ft. above base.

After failure.—Fusion at bottom fluxed away about ½ in. of brick. Brick at bracket had just begun to fuse. (Figs. 76, 131, and 168.)

(*h*) REINFORCED-CONCRETE COLUMNS

TEST NO. 70. SQUARE VERTICALLY REINFORCED; LIMESTONE CONCRETE.—16 mins.—corners of column glowing slightly in lower 3 ft. 34 mins.—column luminous entire length; slight flaking on corners near bottom. 1 hr. 11 mins.—fine vertical crack on east, 2 to 3 ft. up. 1 hr. 28 mins.—similar crack 6 in. long 1 ft. from base. 3 hrs. 2 mins.—similar crack on east, 12 in. long near middle of column. 5 hrs.—maximum expansion. 6 hrs. 39 mins.—fine vertical cracks with ends running horizontally to corners appeared on various faces. 7 hrs. 50 mins.—cracks on east opening slightly; also a few additional fine cracks. 8 hrs.—column still supporting working load with no apparent change; no spalling. 8 hrs. 1 min.—load increased with fire going until failure occurred under 294 000 lbs. about 11 ft. above base, at 8 hrs. 40¼ mins.

After failure.—Concrete near outside calcined, but had hard surface due to partial fusion of the sand in the concrete. (Figs. 76 and 132.)

TEST NO. 71. SQUARE VERTICALLY REINFORCED; TRAP CONCRETE.—20 mins.—column glowing entire length. 2 hrs. 40 mins.—maximum expansion. 3 hrs.—fine vertical cracks on east and west near bottom. 3 hrs. 10 mins.—slight flaking at corners near middle. 4 hrs. 10 mins.—the fine cracks on east and west faces extending in length. 4 hrs. 40 mins.—several additional fine cracks on east and west faces, 4 to 24 in. long. Very little change before failure; no spalling. 5 hrs.—slight deflection to north noted, increasing to ¼ in. at 7 hrs. 22 mins. 7 hrs. 22¾ mins.—failure by compression about 5 ft. above base, reinforcing bars buckling outward.

After failure.—Concrete fused to average depth of 1 in. up to 7½ ft. above base; incipient fusion from 7½ to 9 ft. up, no fusion above. Concrete dry and loose in texture at top. Bars apparently straight, except where buckled at point of failure. Concrete cracked at bars at some points, but in general its condition at the corners was about the same as at the middle of the sides. (Figs. 77, 133, and 169.)

TEST NO. 72. ROUND VERTICALLY REINFORCED; LIMESTONE CONCRETE.—20 mins.—slight surface flaking. 1 hr.—column glowing entire length. 5 hrs.—maximum expansion. 5 hrs. 26 mins.—no spalling or cracking noted. 7 hrs. 55 mins.—a few cracks noted on east and west faces $\frac{1}{16}$ to ⅛ in. wide and 3 to 10 in. long, as follows: On east at 3 and 8 ft. up, on west at 2 and 6 ft. up, and at bracket. 8 hrs.—column still supporting working load with no apparent change; no spalling and but few cracks, as noted; deflection less than ⅛ in. 8 hrs. 2 mins.—load increased with fire going until failure occurred under 250 000 lbs., about 6 ft. above base, at 8 hrs. 4½ mins., reinforcing rods buckling outward.

After failure.—Surface hard immediately after test due to partial fusion of sand. A few days after test concrete flaked off to depth of 1 in. due to calcination of limestone. (Figs. 77, 134, and 170.)

TEST NO. 73. ROUND VERTICALLY REINFORCED. TRAP CONCRETE.—19 mins.—piece of concrete about 8 in. wide and ½ in. deep spalled on west 2 ft. above base. No cracking or other spalling noted before failure. Furnace gases very heavy, making observation difficult; deflection not measured. 4 hrs. 10 mins.—maximum expansion. 7 hrs. 57½ mins.—failure at 2 to 4 ft. up, concrete crushing and shearing on inclined planes; reinforcing bars buckled outward.

After failure.—Concrete fused to depth of 1 in. at break, fusion being more decided in lower than in upper half. Little or no concrete had run. A large number of fine vertical and horizontal cracks present more or less over whole surface of column. (Figs. 77 and 135.)

TEST NO. 74. HOOPED REINFORCED; LIMESTONE CONCRETE.—40 mins.—no spalling or cracking noted. 53 mins.—two fine cracks 3 in. long, on west, 5½ ft. up, opening to $\frac{1}{16}$ in. wide and 10 in. long at 2 hrs. 36 mins. 5 hrs. 50 mins.—maximum expan-

sion. 6 hrs. 50 mins.—several cracks $\frac{1}{16}$ in. wide and about 8 in. long at 2½ ft. up, and one crack $\frac{1}{16}$ in. wide and 8 in. long on east, 5½ ft. up; no cracks above. 8 hrs.—column still supporting working load with no apparent change; no spalling; deflection less than ⅛ in. 8 hrs. 5 mins.—load increased with fire going until failure occurred under 243 000 lbs. about 3 ft. above base, at 8 hrs. 6½ mins.

After failure.—Eight breaks in spiral reinforcement occurred near failure point. Vertical bars buckled out 3 in. Concrete spalled outside of spiral 2 to 4 ft. up; otherwise no plane of cleavage at spiral. (Figs. 77, 136, and 171.)

TEST NO. 75. HOOPED REINFORCED. TRAP CONCRETE.—

First Test.—Gas shut off after 30 mins. to repair burner and test postponed till next day; column apparently not affected.

Second Test.—36 mins.—column glowing dull red for full length. 40 mins.—fine vertical crack, 4 in. long, on southeast 5½ ft. up, increasing to 12 in. long at 1 hr. 45 mins. 1 to 2 hrs.—some six or eight fine cracks, 2 to 12 in. long noted in lower half. 2 hrs. 20 mins.—all cracks opening slightly. 3 hrs.—many very fine cracks on all surfaces. 4 hrs. 50 mins.—maximum expansion. 6 hrs. 25 mins.—cracks opened somewhat; observation difficult. 8 hrs.—column still supporting working load with little apparent change; no spalling; deflection at 8 hrs. less than ⅛ in. 8 hrs. 1½ mins.—load increased with fire going, failure occurring under load of 163 000 lbs., about 3 ft. above base, at 8 hrs. 1¾ mins.

After failure.—Concrete fused to a depth of 1½ in. up to 11 ft. above base; concrete fluxed off to depth of 1 in., 7 to 9 ft. above base. No fusion in upper 1½ ft. due to excess of mortar near surface. Failure apparently due to yielding of spiral, although no break in it occurred. Vertical bars buckled out 1 in. at failure point. Concrete outside of spiral shells off readily. (Figs. 77 and 137.)

NOTE.—For tests Nos. 76 and 77, see under paragraph (*e*) above, hollow clay-tile protections, after test No. 63.

(*i*) TIMBER COLUMNS

TEST NO. 78. LONG-LEAF PINE WITH CAST-IRON CAP AND PINTLE; PROTECTED BY A 1-IN. LAYER OF PORTLAND-CEMENT PLASTER ON METAL LATH.—5 to 8 mins.—vertical cracks in lower half near northwest and southeast corners open to ⅜ in. and 3 to 5 ft. long. 9 mins.—vertical crack at northeast corner at bracket $\frac{3}{16}$ in. wide, 12 in. long. 11 to 40 mins.—finish coat bulging out and spalling in lower half on north and south sides. 16 mins.—crack at bracket on west, ⅛ by 8 in. 20 mins.—crack at pintle, northeast corner, open ¼ in. 44 mins.—⅛ in. vertical crack near northeast corner 1½ to 6½ ft. up. 52 mins.—lath and supporting channel buckle out 1½ in. on south side at west corner, 4 ft. up. 55 mins.—cracks at bracket on northwest corner open ⅛ in., on southwest corner ¼ in. 57 mins.—spurts of flame issue at southwest corner where lath buckled out. 58 mins.—crack at bracket on northeast corner open ⅜ in., on southeast open $\frac{1}{16}$ in. 1 hr. 13 mins.—flames issue from cracks at bracket on west; flames free and full at southwest corner, 4 ft. up. 1 hr. 21 mins.—plaster spalled to lath, 8 by 12 in., on west, 2½ ft. up; flames issue at this point. 1 hr. 24 mins.—flames issue from crack at bracket on northeast corner. 1 hr. 57 mins.—head of column going down quite rapidly; column cap apparently level; crack noted in plaster at base of cap. 1 hr. 58 mins.—plaster spalled to lath, 12 by 12 in. on east, 2 ft. up. 2 hrs. 6 mins.— cap still level. 2 hrs. 8 mins.—little change in plaster during past 50 mins., except as noted. 2 hrs. 9 mins.—small piece of plaster spalled on northwest corner, at lower part of bracket. Settlement of top of column due to heating of cap $\frac{1}{16}$ in. at 1 hr. 10 mins., ¼ in. at 1 hr. 50 mins., 1¼ in. at 2 hrs., and 3¼ in. at 2 hrs. 15 mins. (See Fig. 47.) 2 hrs. 15¼ min.— failure, due to cracking of cap. 2 hrs. 17 mins.—water applied to column extinguishing flames at 2 hrs. 21 mins.

After failure.—Cap broken into three pieces by two transverse cracks across middle portion. A 2 by 2 by ½ in. chip broke off bottom edge of pintle on north. Beam ends charred to depth of 1 in. on sides and ends; inner surface surrounding pintle scarcely charred. Sides of column charred to depth of 1¼ in. in lower two-thirds and to about 1 in. toward top, minimum section of unburned wood, about 8 by 8 in.; vertical cracks on sides extend inward not over ½ in. Top bearing surface of column not charred except for ¼ in. at edges, but was crushed and frayed, the fibers being bent out over 3 in. beyond sound wood at the sides, and pushed up 2 to 3 in. into cracks in cap. A piece was sawed from southwest corner of top and fibers were found to have been crushed and turned over for a depth of 1¾ in. below surface. Wood crushed down on northwest corner 2 in. more than on south side of bearing surface, due to pintle bearing on this side and transmitting the blow from ram at failure Length of column below cap before test, 11 ft. 1¾ in.; average after test, 10 ft. 9 in.; decrease in length, 4¾ in. (Figs. 79, 140, and 47.)

Test No. 79. Long-leaf Pine with Cast-iron Cap and Pintle; unprotected.—2 mins.—surface of column blazing in lower 3 ft. 3 mins.—fine horizontal checks due to charring appeared. 9 mins.—column flaming all over. 11 mins.—three or four vertical cracks beginning to show near middle of each side, $\frac{3}{16}$ in. wide. 18 to 23 mins.—horizontal cracks showing in beam ends at top increasing to ½ in. at 32 mins. 27 mins.—horizontal checks $\frac{3}{16}$ in. wide and 1¼ in. apart quite general. 30 mins.—lazy reddish flames from combustion of wood envelop whole column. 32 mins.—vertical cracks in charred column generally ⅜ in. wide. 42 mins.—head of column going down fast, but no visible sign of distress. 44 mins.—vertical cracks ⅝ in. wide; horizontal checks ¼ in. wide. Settlement of head of column due to heating of cap $\frac{1}{16}$ in. at 25 mins., 1⅜ in. at 40 mins., and $3\frac{1}{16}$ in. at 46 mins. (See Fig. 47.) 50 mins.—failure due to cracking of cap.

After failure.—Cap broken into two pieces, cracking transversely at center. Pintle intact except for slight rounding of lower bearing surface. Beam ends charred to depth of 1½ in. on sides and ends; inner surface surrounding pintle barely scorched. Sides of column charred to depth of 1¼ in.; minimum section of unburned wood 8¾ by 8¾ in. Vertical cracks on sides extend inward to a depth of ¾ in. below sound wood. Column split in two in upper 3 ft. due to end of pintle being forced down through cap into top of column at failure. Top bearing surface very little charred but fibers crushed and broomed, forced outward 3 in. beyond sides, also down into crack in column and up into crack in cap. Fibers tough and ropy and bent over for a length of 3 in. Length of column before test, 11 ft. $1\frac{7}{16}$ in.; average after test, 10 ft. $7\frac{7}{16}$ in.; decrease in length, 6 in. (Figs. 80, 140, and 47.)

Test No. 80. Long-leaf Pine, with Steel-plate Cap and Timber-strut Bearing; column and cap protected by one thickness of ⅜-in. gypsum wall board.—1 min.—kalsomine burned off, showing filler in joints and nailing. 5 mins.—paper on outer surface of wall board charring and flaking, about one-half off at 7 mins. 15 mins.—about two-thirds of outer paper now off; no curling of wall board. 16 to 18 mins.—cracks noted near corners at bracket on east and west sides, up to ⅛ in wide, extending and increasing to ¼ in. on west at 26 mins. 19 to 25 mins.—horizontal cracks developed on north, south, and west faces from 1 to 4 ft. up; wood burning freely at cracks, also a little along corners. 27 mins.—corner beading buckled out at several places, corners burning freely up to 7 ft. above base and to top at 40 mins. 31 mins.—cracks and free burning of wood on east face to 9½ ft. up. 32 mins.—about two-thirds of wall board on west side of cap fell, exposing steel. 35 mins.—board fell at cap on east, exposing all of steel. 37 mins.—board in lower half buckled out at corners; horizontal cracks about 18 in. on centers on all sides with free burning of wood. 41

to 54 mins.—board fell off in places, exposing wood; at end of this period wood was exposed on north at 4 ft. up, on east 2 to 8 ft. up, and on south 2 to 10 ft. up; board remaining in place, much shattered. 1 hr. 3 mins.—continued cracking and falling of board; steel cap slanting slightly to north. 1 hr. 6 mins.—all board down on north 2 to 4 ft. and 7 to 8 ft. up; on west from 4 ft. up to top. 1 hr. 10 mins.—cap tilted, south edge dropped 1½ in. Settlement of top of column due to heating of cap $\frac{1}{16}$ in. at 15 mins., ½ in. at 40 mins., and 2⅞ in. at 1 hr. 10 mins. 1 hr. 13 mins.—failure by tilting of cap, and slipping of same on top bearing strut, column and cap being carried to north.

After failure.—Side plates of cap bent, also inner bolt on south side; bearing plate dished up 1 in. at middle. Strut did not slip and its upper bearing surface was uninjured; sides of strut charred to depth of ½ in., lower bearing surface of strut charred about ¼ in. on edges, brown over rest of area. Fibers crushed and bent over 1½ to 3 in. Strut cracked by longitudinal shearing. Sides of column charred to depth of 1 to 1¼ in.; minimum section of unburned wood, 9⅜ by 8¾ in.; vertical cracks burnt in ¾ to 1 in. deeper. Top bearing surface of column charred only at edges; surface convex, middle being about ⅜ in. higher than edge; fibers crushed and bent over to south for depth of about ½ in.; bearing surface smooth and hard. Length before test, 12 ft. 2 in.; average after test, 12 ft. 1 in.; decrease in length, 1 in. (Figs. 81, 141, and 47.)

Test No. 81. Long-leaf Pine with Steel-Plate Cap and Timber-Strut Bearing; unprotected.—2 mins.—surface of column blazing, lower 3 ft. 6 mins.—columns burning freely on south and west, lower half. 5 mins.—a few vertical cracks near middle. 8 mins.—column charred; irregular surface checks $\frac{3}{16}$-in. wide, quite general. 16 mins.—several vertical cracks ¼ in. wide on all sides; vertical cracks, $\frac{5}{16}$ in. wide. 19 mins.—gases heavy, making observation difficult. 32 mins.—color noted in cap. 34 mins.—all cracks about ⅜ in. wide; settlement of top of column due to heating of cap $\frac{1}{16}$ in. at 20 mins., increasing to $\frac{1}{16}$ in. at 30 mins., and $1\frac{9}{16}$ in. at 34 mins. 35 mins.—failure with top of column sliding to south and west; fire kept burning until 43½ mins., as it was not possible to ascertain definitely if failure had taken place, owing to heavy smoke; column yielded quite gradually, only slight report heard at failure. 45 mins.—water applied to column extinguishing flames completely at 55 mins.

After failure.—Upper part of side plates of cap bent toward the east; bearing plate dished up ¾ in. at middle. Strut did not slip and its upper bearing surface was uninjured; lower bearing surface of strut charred to depth of ⅛ in. and fibers crushed and bent over to depth of ½ to 1 in.; strut split by longitudinal shearing. Sides of column charred to depth of $1\frac{5}{16}$ in.; minimum section of unburned wood 9 by 9 in. Top bearing surface of column charred not deeper than ⅛ in. except at edges; fibers crushed and bent over to north for a depth of about ¼ in. Length before test, 12 ft. 2 in.; average after test, 12 ft. 1⅛ in.; decrease in length, ⅞ in. (Figs. 81, 141, and 47).

Test No. 82. Douglas Fir with Cast-Iron Cap and Pintle; unprotected.—3 mins.—surface of column blazing in lower 3 ft.; horizontal surface checks due to charring, 1½ in. on centers. 10 mins.—column blazing freely all over; several vertical checks on all sides, $\frac{3}{16}$ in. wide, small pieces of charcoal falling. 14 mins.—horizontal checks $\frac{3}{16}$ in. wide. 24 mins.—horizontal and vertical checks about ¼ in. wide. 25 mins.—column flaming all over, but not so freely as pine columns at this stage. 36 mins.—pieces of charcoal falling off under cap, indicating rapid dropping of head of column. 38 mins.—cap nearly level; no color. 41 mins.—buckling under cap more pronounced; noise heard at bearing. 42½ mins.—cap still nearly level. 44 mins.—wood fibers under cap appear to crush and buckle out. Settlement of top of column due to heating of cap $\frac{1}{16}$ in. at 15 mins., 1 in. at 30 mins., 3½ in. at 41 mins., and 5⅛ in. at 45 mins. 45¼ mins.—failure due to cracking of cap.

After failure.—Cap broken transversely into two pieces. Small pieces chipped from outer edge of pintle on southwest at bottom; otherwise pintle uninjured. Beam ends charred on sides and ends to maximum depth of 1 in.; inner surface of wood surrounding pintle very slightly charred in places. Sides of column charred to depth of $1\frac{1}{16}$ in.; minimum section of unburned wood, $9\frac{1}{2}$ by $9\frac{3}{8}$ in.; vertical cracks did not extend into uncharred wood. Top bearing surface of column charred not more than $\frac{1}{8}$ in. deep, but was broomed and crushed; fibers bent out beyond sides of column, also pushed up 3 in. into crack in cap. A piece cut out of southwest corner showed fibers crushed and turned over 3 to $3\frac{1}{4}$ in. below bearing surface; fibers tough, hardly scorched. Length before test, 11 ft. $1\frac{5}{8}$ in.; average after test, 10 ft. 6 in.; decrease in length, $7\frac{5}{8}$ in. (Figs. 82, 140, and 47.)

Test No. 83. Douglas Fir, with Steel-Plate Cap and Timber-Strut Bearing; unprotected.—$2\frac{1}{2}$ mins.—surface of column blazing lower half; top just beginning to burn. 2 to 4 mins.—crackling noises heard. 5 mins.—column blazing all over. 15 mins.—very little blazing, hardly any crackling, secondary air gate found shut; opened. 18 mins.—horizontal surface checks $\frac{1}{8}$ in. wide, 1 in. on centers; 2 to 4 vertical cracks on each side not over $\frac{1}{8}$ in. wide. 24 mins.—column again burning freely. 31 mins.—horizontal checks $\frac{1}{4}$ in. wide. 37 mins.—vertical cracks up to $\frac{3}{16}$ in. wide; very little crackling. 38 mins.—cap tilted, south edge dropped $1\frac{1}{2}$ in.; settlement of top of column, due to heating of cap $\frac{1}{16}$ in. at 14 mins., $1\frac{1}{8}$ in. at 30 mins., and $2\frac{9}{16}$ in. at 38 mins. $38\frac{1}{2}$ mins.—failure with top of column sliding to north carrying cap with it. $40\frac{1}{2}$ mins.—water applied to column extinguishing flames completely at $45\frac{1}{2}$ mins.

After failure.—Side plates of cap bent outward at top, also inner bolt on north bent out; bearing plate dished up 1 in. at middle in longitudinal direction. Strut did not slip, and its upper bearing surface was uninjured; lower bearing surface of strut charred to depth of $\frac{1}{8}$ in. and fibers crushed and bent over to north maximum of 1 in. on north side. Sides of column charred to depth of $\frac{15}{16}$ in.; minimum section of unburned wood $9\frac{1}{2}$ by $9\frac{1}{2}$ in.; top of bearing surface of column very little charred, but discolored to depth of 1 in.; surface convex, rounded over from $\frac{1}{2}$ to 1 in. at edges; fibers crushed and bent over mostly on east and west sides; wood quite soft to a depth of $\frac{1}{4}$ in. Length before test, 11 ft. 2 in.; average after test, 11 ft. $1\frac{1}{4}$ in.; decrease in length, $\frac{3}{4}$ in. (Figs. 82, 141, and 47).

XI. RESULTS OF FIRE AND WATER TESTS

1. APPLIED LOADS, DURATION AND EFFECT OF FIRE AND WATER

In the fire and water test, the column was subjected to working load, and to fire for a predetermined period not exceeding one hour, after which a hose stream was applied to three sides. On cooling, the column was either loaded to failure or subjected to an excess load equal to about twice the load sustained during the fire and water periods.

A general summary of applied loads, duration and relative intensity of fire exposure, duration and pressure of hose-stream application, and the general effects of fire and water are given in Table 44. Further details of columns and protections are given in Tables 4*a* and 4*b* (p. 51–54).

2. PHOTOGRAPHIC RECORDS

Views of the columns in the fire and water series at several stages of the test are given in Figs. 83 to 89, Appendix A (p. 242–248).

3. FURNACE AND COLUMN TEMPERATURES

The temperatures observed in the furnace and test columns are given by the curves in Figs. 142 to 145, Appendix B (p. 303–306). The arrows on the plots indicate the time water was applied in each test.

4. LONGITUDINAL DEFORMATION

The expansion of the column during the fire period and contraction on application of water were determined by measurement of the movement of the head of the column, and the amounts are given in the respective test logs. These effects were quite small except in the case of the unprotected cast-iron columns, both of which attained maximum expansion before water was applied (Fig. 46, p. 124).

5. SUBSEQUENT LOADING TESTS

The loading to which the columns were subjected subsequent to the fire and water test are given in Table 44. They are loaded to their maximum sustaining capacity with the exception of four columns that were loaded to about twice their design working load and reserved for use in further tests, and the one protected by plaster on metal lath on which a subsequent fire test to failure under working load was made.

6. LOG OF FIRE AND WATER TESTS

(*a*) CONCRETE PROTECTIONS

TEST NO. 101. ROLLED H. 2-IN. CONCRETE PROTECTION. UPPER SECTION, LIMESTONE; MIDDLE SECTION, TRAP; LOWER SECTION, JOILET GRAVEL. TIED.

Fire Test.—30 mins.—no effect noted; flame very smoky. 32 mins.—vertical cracks on east and west faces near corners in lower 3 ft., ¼ in. wide at bottom. 38 mins.—⅛-in. vertical crack at center of south face, 2 ft. up. 45 mins.—cracks open $\frac{3}{16}$ in. to ⅜ in.; small spall on southwest corner, 3 ft. up. 60 mins.—gas shut off; no change in column except cracks had opened up slightly.

Water Test.—61¼ mins.—water applied to column through a 1⅛-in. nozzle; pressure at base of play pipe, 50 lbs. per sq. in.; nozzle was maintained at a distance of 20 ft. away from and to the west of column in all tests and the stream applied directly to the west face and at an angle to the north and south of faces; stream was played slowly up and down over all three faces of column. 61½ mins.—pieces washed off southwest corner and west face, 3 ft. up. 61¾ mins.—some pieces fell off west face in upper section. 62¼ mins.—pieces falling on west face and corners in middle section. 63¾ mins.—concrete fell in lower 2 ft., exposing steel. 64¼ mins.—north

face washed off to depth of 1½ in., 1 to 2½ ft. up; edge of flange exposed on northwest corner. 65¾ mins.—pieces fell off north face near top. 66¼ mins.—water shut off; duration of water test, 5 mins.

After Test.—Edge of flange exposed on northwest corner 1 to 2½ ft. up, and southwest corner for 4 in. at 5 ft. and at 8 ft. up; these corners spalled not over 1½ by 1½ in. for rest of length. Concrete in center of west face generally pitted from ⅛ to ¼ in. for the whole length; at bottom of upper section concrete pitted to depth of 1 in. for about 6 in. On south face all concrete washed off in lower 3 ft., exposing flange; about two-thirds of rest of surface pitted slightly. On north face surface generally smooth except at northwest corner. Fine vertical cracks in east face nearly full length, 1½ in. from corners. Column expanded 0.26 in. during the fire test and contracted 0.06 in. during the water test.

Load Test.—On the following day the column was subjected to a load of 234 000 lbs. this being calculated as the design dead load plus 2½ times the live load, assuming the former to be one-third of the latter. A faint cracking sound was heard at 69 000 lbs. At about 175 000 lbs. a dull thud was heard and several small pieces of concrete fell off northwest corner about halfway up. Column withstood test satisfactorily. The column recovered from the depression due to the loading within 0.002 in. on release of load. (Figs. 83 and 142.)

TEST NO. 102. ROLLED H. 2-IN. CONCRETE PROTECTION. UPPER SECTION, TRAP; MIDDLE SECTION, JOILET GRAVEL; LOWER SECTION, LIMESTONE. NO TIE.

Fire Test.—21 mins.—slight flaking at southeast corner, 9 ft. up. 41 mins.—column glowing all over. 60 mins.—gas shut off. No cracking or spalling.

Water Test.—62¼ mins.—water applied to column, 50-lb. pressure. 62¾ mins.—cracks noted on west face near both corners; steam very heavy, making observation difficult. 64 mins.—concrete fell on north in lower 10 ft., exposing steel. 65 mins.—long vertical crack at southwest corner, exposing flange in places in lower 10 ft. 67¼ mins.—water shut off; duration of water test, 5 mins.

After Test.—North flange exposed completely 1½ to 11 ft. up. South flange exposed on southwest corner for 2½ by 14 in. at 2½ ft. up, for 4 by 20 in. at 5 ft. up, for 3 by 10 in. at 8 ft., for 1¼ by 12 in. at 10 ft. up; concrete loose on south flange from 1 to 11½ ft. up. On west concrete washed off to depth of ½ to 1½ in. at 3 ft., 4½ ft., 8 ft., and 10½ ft. above base; concrete least washed off on middle section; column expanded 0.26 in. during the fire test and contracted 0.09 in. during the water test.

Load Test.—On the following day the column in the unstripped condition was loaded until failure occurred at 442 000 lbs. by buckling to west, about 8 ft. above base. Scaling on the surface of the steel was first noted on the north flange at 373 000 lbs., the yield point of the steel as determined from the depression curve of the top of the column, being attained at 400 000 lbs. (Figs. 83 and 142.)

TEST NO. 103. PLATE AND ANGLE. 4-IN. CONCRETE PROTECTOIN. UPPER SECTION, TRAP; MIDDLE SECTION, GRANITE; LOWER SECTION, LIMESTONE. TIED.

Fire Test.—60 mins.—gas shut off; $\frac{1}{16}$-in. vertical crack, 4 in. long, on west near southwest corner at bracket; no other cracking, spalling or other visible effects on column.

Water Test.—62 mins.—water applied to column; 50-lb. pressure. 62¾ mins.—small piece falling on west; face and corners rounded. 63½ mins.—continued falling of small pieces; steam heavy. 64½ min.—water effects most marked at bottom of middle section. 65½ mins.—continued falling of small pieces. 67 mins.—water shut off; duration of water test, 5 mins.

After Test.—Little effect on north and south faces back of corners except in middle section, where granite concrete was pitted to depth of ¼ in. quite generally. On west face the middle section was most affected and the upper section least affected. concrete in middle section quite generally washed away to depth of 1½ in. at center, and 3½ in. at corners; in upper and lower sections, ¼ to ½ in. at centers and 3½ in. at corners; concrete in middle section cracked along flanges near each corner; steel flanges not exposed. East face unaffected. Column expanded 0.15 in. during the fire test.

Load Test.—On the following day the column in the unstripped condition was loaded to 547 000 lbs., the capacity of the machine, without signs of distress or failure. Column was then stripped from 5 ft. 2 in. to 7 ft. 10 in. above base and loaded until failure occurred at 537 000 lbs. by buckling to west, 7 ft. 8 in. above base. Scaling of steel was first noted at 433 000 lbs. and decided yielding began at 475 000 lbs. (Figs. 83 and 142.)

TEST NO. 104. PLATE AND ANGLE. 2-IN. CONCRETE PROTECTION. UPPER SECTION, CINDER; MIDDLE SECTION, SANDSTONE; LOWER SECTION, TRAP. TIED.

Fire Test.—31 mins.—fine vertical crack, 24 in. long, on west near southwest corner, 6½ ft. up; also on east, southeast corner, 4½ ft. to 7 ft. up; slight crushing on southeast corner 8 ft. up, small piece falling at 33 mins. 35 mins.—¼-in. crack on center of south face, 6½ to 9½ ft. up, extending to southwest corner at upper end. 50 mins.—cracks previously noted opening up ¼ to ½ in. and extending in length in middle section. 53 mins.—piece spalled off northeast corner, 7 ft. up. 60 mins.—gas shut off; only cracks are the three noted in middle section, 3 to 4 ft. long, and ¼ to ½ in. wide; no cracks on north.

Water Test.—61½ mins.—water applied to column; 50-lb. pressure. 61¾ mins.—concrete fell off on south 5 to 8 ft. up, exposing steel. 63 mins.—concrete in lower 4 ft. on west washed and pitted considerably. 63½ mins.—concrete on northwest corner fell, 4 to 7 ft. up. 64¾ mins.—upper 4 ft. on west pitted. 65 mins.—piece fell on northwest corner, 10 ft. up. 66 mins.—pieces washed off corners on west in lower 4 ft. 66½ mins.—water shut off; duration of water test, 5 mins.

After Test.—West face pitted ½ to 2 in. in lower section, ¼ in. in middle section, and ½ to 2 in. in upper section; northwest corner off for full length, exposing flange edge for 1½ by 12 in. at 2 ft. up, for 2 by 12 in. 4 ft. up, and for 1½ in. 5 to 9 ft. up; north face intact except for spalling on northwest corner; southwest corner spalled full length exposing flange edge for 3 by 4 in. 2 ft. up, and 2 by 6 in. 6 ft. up; entire south flange exposed in middle section; east side intact except for a few fine cracks. Column expanded 0.26 in. during the fire test and contracted 0.09 in. during the water test.

Load Test.—On the following day the column in the unstripped condition was subjected to an excess load of 234 000 lbs. without developing any signs of distress and with full recovery from the deformation on removal of load. (Figs. 84 and 142.)

(*b*) HOLLOW CLAY-TILE PROTECTIONS

TEST NO. 105. PLATE AND ANGLE. 2-IN. HOLLOW CLAY-TILE PROTECTION, UNFILLED. UPPER SECTION, SURFACE CLAY, BOSTON DISTRICT; MIDDLE SECTION, SEMIFIRE CLAY, NEW JERSEY DISTRICT; LOWER SECTION, OHIO SHALE. NO FILLING. OUTSIDE WIRE TIES.

Fire Test.—2 to 3 mins.—vertical cracks and spalling of parts of outer shells on north and west sides in lower section. 3 to 20 mins.—continued cracking, spalling, and bulging of outer shells in lower section; tile in upper half very little affected. 30 mins.—

a little more spalling in lower section; vertical cracks on west at bracket, and at south and west 5 ft. up, ⅛ in. wide. 39 mins.—outer shells spalled off on east and west, 3d course, shell hanging loose on east, 2d course. 45 mins.—gas shut off; very little damage to middle and upper sections except for vertical cracks as noted; lower three courses cracked and spalled.

Water Test.—46 mins.—water applied to column; 30-lb. pressure. 46½ mins.—all tile washed off on west 1 to 2 ft. up, and on north 1 to 3 ft. up, exposing steel. 47 mins.—all tile down in lower sections on north and west 1 to 4 ft. up. 47½ mins.—tile down on north and west 4 to 5 ft. up. 48 mins.—tile still in place on south except outer shell spalled 3 to 4 ft. up; tile down on west in center 5 to 6 ft. up. 48½ mins.—water shut off; duration of water test, 2½ mins.

After test.—Tile down and steel exposed on north and east 1 to 5 ft. up and on west 1 to 6 ft. up; tile loose on south 1 to 5 ft. up. Tile in upper half of column almost intact except outer shell cracked on west 6 to 7 ft. up, and part of outer shell off on north at bracket. Ties in lower five courses fell down, other ties in place. Expansion during fire test, 0.22 in.; contraction during water test, 0.13 in.

On the following day the column was stripped. Mortar in joints found to be fairly full, although in about one-half of the joints it had been washed out to depth of ½ to ⅝ in. Mortar on flanges broke away from the steel and adhered to the tile. Lateral deflection less than ⅛ in.

Load Test.—The stripped column was loaded until failure occurred at 445 000 lbs. with buckling to east about 6 ft. above base. (Figs. 84 and 143.)

TEST NO. 106. PLATE AND ANGLE. 2-IN. HOLLOW CLAY-TILE PROTECTION. UPPER AND LOWER SECTIONS, OHIO SEMI-FIRE CLAY; MIDDLE SECTION, SURFACE CLAY, CHICAGO DISTRICT. OUTSIDE WIRE TIES IN UPPER HALF OF COLUMN, WIRE MESH IN JOINTS IN LOWER HALF. CONCRETE FILLING.

Fire Test.—2½ to 6 mins.—outer shells spalling at corners in middle section. 9 to 12 mins.—outer shells shattered on west in middle section, but did not spall off; ⅛-in. cracks near corners on west in upper section. 12½ to 18½ mins.—outer shells spalled and bulging in middle section; at end of this period, all of outer shells were down in 5th course on north and west and partly down on east. 20½ mins.—vertical crack in upper section on west 9½ to 11½ ft. up, ⅝ in. wide, opening to 1 in. at 26 mins. 23 mins.—outer shell buckled on east, 9 ft. up. 25½ mins.—tie wire broken on 10th course. 31 to 36 mins.—continued cracking and spalling of outer shells in middle section, also bulging out on north 7 ft. up. 38 mins.—outer shell spalled on 6th course on north. 40 mins.—all outer shells spalled on east 3 to 5 ft. up. 44 mins.—outer shells on east 7 ft. up buckle out 2 in. 45 mins.—gas shut off; outer shells are off on 5th, 8th, and 9th courses on west, and on 5th and 6th courses on north; column expanded 0.24 in. during the fire test.

Water Test.—47½ mins.—water applied to column; 30-lb. pressure. 47¾ mins.—considerable amount of tile fallen; steam obscures view. 48½ mins.—tile down in middle 8 ft. on north, south and west, exposing flanges on north and south; concrete filling in place. 49¾ mins.—water off; duration of water test, 2¼ mins.

After Test.—Steel exposed on north 1 to 11 ft. up, on south from 1 to 10 ft. up except where parts of inner shells remained in place; wire mesh still in place on north at 1 ft., 4 ft., 5 ft. and 6 ft. up. On west tile off, exposing concrete 2 to 10 ft. up; outer shell off 10 to 12 ft. up. On east side, concrete exposed 2 to 4 ft. up; the rest was covered by tile or inner shells of same. Tile all in place on bottom course on all sides and on bracket course on north, east, and south sides.

Load Test.—On the following day the column was subjected to an excess load of 228 000 lbs. without developing any signs of distress and recovered from deformation almost completely on removal of load. (Figs. 85 and 143.)

TEST NO. 107. PLATE AND CHANNEL. 4-IN. HOLLOW CLAY-TILE PROTECTION. UPPER SECTION, OHIO SHALE; MIDDLE SECTION, SEMIFIRE CLAY, NEW JERSEY DISTRICT; LOWER SECTION, SURFACE CLAY, BOSTON DISTRICT. OUTSIDE WIRE TIES ON UPPER HALF OF COLUMN; WIRE MESH IN JOINTS IN LOWER HALF. NO FILLING.

Fire Test.—12 mins.—no cracking or spalling noted. 15 mins.—vertical cracks on east from 8 ft. up to top, ¼ in. wide; part of outer shell spalled on 11th course, east; $\frac{1}{16}$-in. vertical crack on east 6 to 8 ft. up. 19 mins.—$\frac{1}{16}$-in. vertical crack on west 6 to 9 ft. up. 22 mins.—cracks on east and west near top, ½ to 1 in. wide. 25 to 27 mins.—outer shells on west bulging out 1½ to 2 in. in upper section, held by ties. 31 to 35 mins.—¼-in. vertical crack on west at south corner from 9 ft. up to top; other cracks opening up and outer shells bulging in upper section. 42 mins.—continued cracking in upper section; new crack on north 1 in. wide, 11th course; tile on east 11th and 12th courses cracked. 45 mins.—gas shut off; tile in upper section generally cracked and outer shells loose; lower section practically intact; middle courses have a few vertical cracks as noted.

Water Test.—46 mins.—water applied to column; 30-lb. pressure. 46½ mins.—little effect; outer shell on west fractured in upper section. 46¾ mins.—little apparent change. 47¾ mins.—outer shell down on west 9 to 11 ft. up. 48¼ mins.—tile almost all down on upper three courses. 48½ mins.—water shut off; duration of water test, 2½ mins.

After Test.—All tile down in 10th, 11th, and 12th courses, except on north side, where tile is held by bracket; some mortar adheres to north and south flanges. Tile shattered on east and west, in 9th course. Tile in good condition in lower 8 ft. except for a few ⅛-in. vertical cracks on east and west 6 to 8 ft. up, and small hole in outer shell on west 7 ft. up. Mortar and mesh in joints in lower courses in good condition, it being necessary to wedge the tile apart in order to remove the covering. Column expanded 0.15 in. during the fire test and contracted 0.10 in. during the water test. Maximum deflection after fire and water test, $\frac{5}{32}$ in. to west.

Load Test.—On the following day the column was stripped and loaded to failure at 348 000 lbs., column buckling to west about 6 ft. above base. Decided yielding of the steel began at 322 000 lbs. (Figs. 85 and 143.)

(*c*) GYPSUM-BLOCK PROTECTIONS

TEST NO. 108. ROLLED H. 2-IN. SOLID GYPSUM-BLOCK PROTECTION. UPPER HALF, WESTERN GYPSUM; LOWER HALF, EASTERN GYPSUM. FILLED.

Fire Test.—13 mins.—fine surface checks beginning to develop. 28 mins.—surface checks generally ⅛ in. wide and ⅜ in. deep near lower end of column to very fine at top. 36 mins.—joints at bottom open $\frac{1}{16}$ to ⅛ in. 43 mins.—very little change in fire effects. 45 mins.—gas shut off.

Water Test.—46 mins.—water applied to column; 30-lb. pressure. 47¾ mins.—little effect noted except for surface erosion on west. 48 mins.—blocks fell on west 2 ft. 10 in. to 5 ft. 8 in. up. 48¼ mins.—blocks down on west 1½ to 9 ft. up, exposing flange edges and filling. 48½ mins.—blocks down on south, 4¼ to 7 ft. up. 48¾ mins.—blocks down, exposing steel nearly full length on north and south. 49 mins.—water off; duration of water test, 3 mins.

After Test.—All blocks down on north 1½ to 9 ft. up, exposing steel; on south, from 1½ ft. up to top of bracket, exposing steel except for a 2 ft. length above middle of column, where mortar remained in place; on west, 1½ to 4¼ ft., 6 to 9 ft., and from 11 ft. to top. Filling washed out on west, exposing web from 2 ft. 4 in. to 3 ft. up. All blocks in place on east except at places on corners. Blocks remaining on column on west washed off to a depth of ¾ in. to point where fiber in blocks was not charred.

Mortar on flanges, and filling in web had evidently been quite full. Steel not rusted except where exposed by water. Column expanded 0.05 in. during the fire test and contracted 0.05 in. during the water test. Maximum lateral deflection after fire and water tests, $\frac{1}{16}$ in.

Load Test.—On the following day the column was loaded to failure at 311 000 lbs. column buckling to west about 7 ft. above base. (Figs. 86 and 143.)

TEST NO. 109. ROLLED H. 4-IN. SOLID GYPSUM-BLOCK PROTECTION. UPPER SECTION, EASTERN GYPSUM; LOWER SECTION, WESTERN GYPSUM.

Fire Test.—16 mins.—a few corner cracks developed in lower half, $\frac{1}{16}$ by 1 in. 22 mins.—fine surface checks beginning to develop near bottom. 42 mins.—surface checks quite general, forming in $\frac{3}{8}$-in. squares, checks $\frac{1}{32}$ in. wide at bottom and $\frac{1}{64}$ in. wide at top; corner cracks opening to maximum of $\frac{1}{8}$ in.; vertical joints in lower half open $\frac{1}{32}$ in.; slight spalling at corners. 53 to 57 mins.—surface checks open from $\frac{1}{8}$ in. at base to $\frac{1}{32}$ in. at top of column. 59 $\frac{1}{2}$ mins.—gas shut off; surface checked all over, most at bottom; otherwise covering and column little affected.

Water Test.—60¾ mins.—water applied to column; 50-lb. pressure. 61¾ mins.—hose burst, water off; gypsum washed off on west to depth of about 1 in., mortar in joints washed out from 1 to 4 in. depth, exposing ends of ties. Gypsum washed off on north and south sides, ¾ to 1 in. depth; east side intact. 72½ mins.—water again applied; 50-lb. pressure. 73½ mins.—outer 1 in. of gypsum washed off clean on north and south. 74 mins.—almost all of mortar washed out of vertical joints on west. 76¼ mins.—$\frac{1}{16}$-in. vertical crack on west near south corner 9 to 10 ft. up. 76½ mins.—water shut off; total duration of water test, 5 mins.

After Test.—On west side surface washed very smooth, corners rounded; blocks slightly less than 3 in. thick; no mortar in vertical joints; ties all in place and hold mortar in horizontal joints; on north and south sides, faces of blocks rough, with fibers projecting; blocks about 3 in. thick; all joints generally full; east side little affected; no blocks loose; edges of bracket and stiffner angles exposed. Column expanded 0.04 in. during the fire test; less than 0.01 in. contraction during the water test.

Load Test.—On the following day the column in the unstripped condition was subjected to an excess load of 234 000 lbs., which it withstood without any signs of distress except for a few small vertical cracks in the blocks. (Figs. 86 and 143.)

(*d*) PLASTER ON METAL-LATH PROTECTION

TEST NO. 110. PLATE AND ANGLE. TWO LAYERS OF PORTLAND-CEMENT PLASTER ON METAL LATH WITH ¾-IN. AIR SPACE BETWEEN LAYERS.

Fire Test.—6 mins.—furnace gases heavy, difficult to see column. 18 mins.—fire more luminous. 19 mins.—plaster cracked and spalled on all sides at bottom of bracket; this probably occurred at about 14 mins.; corners spalled slightly on west 32 mins.—fine crack on east, 2 in. long, 4 ft. up. 45 mins.—gas shut off; several fine cracks noted on all faces.

Water Test.—46¼ mins.—water applied to column; 30-lb. pressure. 46¾ mins.—vertical cracks near corners on all faces about 6 ft. long. 47¼ mins.—piece of plaster, 8 in. square, spalled off north face, 6 ft. up. 48 mins.—plaster washed off southwest corner in upper and lower 3 ft. 48¾mins.—plaster washed off northwest corner, lower 4 ft. 49½ mins.—water off; duration of water test, 3¼ mins.

After Test.—On west side plaster off, 1 by 1 in., exposing lath on northwest corner 1 to 4 ft. up. Fine horizontal cracks across face at 4 ft., 6 ft., 7 ft., 7½ ft., 8 ft., 9½ ft., and 11 ft. up. On north side, lath exposed near west corner, 5 by 18 in. 6 ft. up. On south side, lath exposed on southwest corner, 1 by 2 in., from 7 ft. up to top; fine

horizontal cracks across face at 5 and 8 ft. up. On east side, vertical crack runs full length near south corner; fine horizontal cracks at 2½, 4, and 6 ft. up. Plaster cracked and lath exposed on all sides at bottom of bracket. Column expanded 0.14 in. during the fire test. No contraction during the water test (see Fig. 46).

Subsequent Fire Test.—Since the protection was little injured by the fire and water test, it was subjected to a second fire test to failure on the day following the fire and water test. 5 mins.—plaster bulging out 2 in. on north at bottom of bracket. 7 mins.—crack on west at southwest corner open ½ in., 7 to 8 ft. up, extending further 4 ft. vertically at 11 mins. 14 mins.—bulging of plaster on southeast below bracket. 39 mins.—plaster bulging away from lath in places on southwest corner from 7 ft. up to top. 46 mins.—northeast corner cracks opening up, 8¼ to 11 ft. up. 1 hr.—bulging of plaster increasing on all faces at bracket. 1 hr. 54 mins.—no change except all cracks opening up slightly. 2 hrs. 25 mins.—about one half of lath exposed at bracket on west. 2 hrs. 28 mins.—small areas of lath exposed at bracket on east. 2 hrs. 47¼ mins.—failure with buckling to west, maximum at 7½ ft. above base. Column expanded maximum of 0.92 in. at 2 hrs. 30 mins.; from this point to one minute before failure it compressed ½ in. (Figs. 87, 144, and 46.)

(*e*) REINFORCED-CONCRETE COLUMNS

TEST NO. 111. SQUARE VERTICALLY REINFORCED CONCRETE. UPPER SECTION, LIMESTONE; MIDDLE SECTION, MERAMEC RIVER GRAVEL; LOWER SECTION, LIMESTONE.

Fire Test.—14 to 24 mins.—slight spalling of small pieces from corners in middle section. 26 to 31 mins.—cracking of concrete in middle section on all sides, cracks $\frac{1}{16}$ to $\frac{3}{16}$ in. wide, and 12 to 30 in. long; cracking and crushing at southwest corner 6½ ft. up, cracks extending diagonally downward on both south and west faces. 36 mins.—upper and lower sections apparently unaffected. 36½ mins.—crack on south 4½ ft. up, extending downward to 2 ft. above base. 37 mins.—cracks at southwest 6½ ft. up, now open ⅜ in. 46 min.—same crack open 1½ in., new crack extending upward 12 in. from same; crack on east, 5¼ ft. up open ⅜ in. 48 mins.—small vertical and diagonal cracks extending from crack at southeast corner 4½ to 5½ ft. up. 53 mins.—$\frac{1}{16}$ in. vertical and diagonal cracks near northwest corner on north and west 5 ft. up. 60 mins.—gas shut off; very little spalling; concrete at southwest corner in middle section, crushed.

Water Test.—61½ mins.—water applied to column; 50-lb. pressure. 63 mins.—steam heavy; observations impossible. 63¼ mins.—concrete spalled to reinforcing rod on northwest corner in lower half. 63½ mins.—west face washed off to about 1½-in. depth. 64 mins.—northwest rod exposed to 10 ft. up. 64½ mins.—southwest rod exposed to 11 ft. up. 65 mins.—concrete washed off in middle of column on west, exposing tie. 65¾ mins.—little change during past ¾ min. 66½ mins.—water off; duration of water test, 5 mins.

After Test.—Concrete washed off both corners of west face 4 by 4 in., exposing reinforcing rods 1 to 11 ft. up; in center of west face concrete is washed off to average depth of 1 in. in upper section, 2 in. in middle section, exposing ties, and ⅛ in. in lower section; on north, east, and south faces, concrete in place in upper and lower sections except at west corners; in middle section, concrete washed off south face to depth of 1 in., 4½ to 5½ ft. up, and off north face to depth of ½ in. in places from 4 to 6 ft. up; east side unaffected by water, but has several vertical cracks in middle section from the fire test. Estimated minimum area of cross section of concrete not spalled or loose in middle section, 187 sq. in., a decrease of a little more than 25 per cent of the original section. Column expanded 0.28 in. during the fire test and contracted 0.09 in. during the water test.

Load Test.—On the following day the column was loaded to failure at 379 000 lbs.; column failed by diagonal shearing from a point 4½ ft. up on southeast corner to a point 8½ ft. up on northwest corner; no cracking or yielding was noted before the maximum load was applied. (Figs. 88 and 145.)

TEST NO. 112. ROUND VERTICALLY REINFORCED CONCRETE. UPPER SECTION, LIMESTONE; MIDDLE SECTION, MERAMEC RIVER GRAVEL; LOWER SECTION, JOLIET GRAVEL.

Fire Test.—18 mins.—flaking on southeast corner, 6 ft. up. 23 to 25 mins.—flaking in middle section on south and east. 30 mins.— concrete crushed on south, 6 to 7½ ft. up, cracks opening to ½ in. at 32 mins.; loose pieces ready to fall. 31 mins.—vertical crack on southwest ¼ in. wide, 4 to 7½ ft. up. 39 mins.—crack on south, 6 to 7 ft. up, open ⅜ in. 43 mins.—several new cracks ⅛ in. wide and about 12 in. long in middle section. 51 to 57 mins.—spalling in middle section to depth of 1½ in.; upper and lower sections intact. 60 mins.—gas shut off.

Water Test.—61¼ mins.—water applied to column; 50-lb. pressure. 62¼ mins.—concrete washed off, exposing tie on west, middle section. 62½ mins.—some concrete fell on south, middle section. 63½ mins.—concrete down on south, 2 to 5 ft. up. 64 mins.—reinforcing bars exposed on west in lower 7 ft. 64½ mins.—piece fell on west, 7 ft. up. 66 mins.—piece fell on south, 8 ft. up; not much change during past 2 mins. 66¼ mins.—water off; duration of water test, 5 mins.

After Test.—Concrete off to reinforcing bars from 3 to 8½ ft. up on west and from 3 to 8 ft. up on south; also piece off, 1 by 6 by 12 in., at 12 ft. up, and 1 by 4 by 5 in., at 11 ft. up on west. On north concrete loose to outside of ties, 2 to 6 ft. up and piece off, 1½ by 12 by 6 in., at 6½ ft. up; on east, crack ½ in. wide from 6 to 9½ ft. up; concrete shattered near middle; crack on northeast corner, ¼ in. wide, 1½ to 4½ ft. up; several smaller horizontal and vertical cracks; section at middle of column reduced about 30 per cent due to spalled or loose concrete. Column expanded 0.14 in. during the fire test and contracted 0.02 in. during the water test.

Load Test.—On the following day, the column was loaded to failure at 423 000 lbs.; failure occured by diagonal shearing from 5 ft. up on north to 7 ft. up on south; cracking sounds were heard at about 290 000 lbs., which increased as failure was approached. (Figs. 88 and 145.)

TEST NO. 113. HOOPED REINFORCED CONCRETE. UPPER SECTION, TRAP; MIDDLE SECTION, MERAMEC RIVER GRAVEL; LOWER SECTION, GRANITE.

Fire Test.—9 to 20 mins.—flaking and cracking in middle section, cracks ¼ in. maximum, 2 to 3 ft. long. 21 mins.—piece ½ by 4 by 4 in. spalled on southwest, 5 ft. up. 22 mins.—crack on southwest in middle section open ¾ in. 23 mins.—flaking and buckling of concrete shell general in middle section. 27 mins.—piece 1 by 4 by 10 in. spalled on southwest, 4½ ft. up. 28 mins.—piece spalled, 1¾ by 10 in. on south 4 to 7 ft. up. 34 mins.—buckling of concrete shell in middle section to 2 in. maximum. 37 mins.—upper and lower sections intact except for cracks extending from middle section. 38 mins.—spalling on south, 4 to 8 ft. up; spiral exposed for 8 turns on 10-in. width. 43 mins.—concrete shell on west middle section, standing out 2 in. 43½ mins.—spalling on east, 4 to 8 ft. up; spiral exposed for 16 turns on 5-in. width. 46 mins.—cracks on west extending down to 1 ft. from base, are ½ in. wide at 3 ft. up, opening to 1½ in. at 52 mins. 58 mins.—cracks on east in middle section, extending down to 3 ft. above base; finer cracks below. 60 mins.—gas shut off.

Water Test.—61¼ mins.—water on, 50-lb. pressure. 61½ mins.—hose burst; water shut off; concrete stripped to spiral on west from 1 to 8½ ft. up and partly on north and south 2 to 6 ft. up; above this on north and south concrete off to within 1 in. from spiral; aggregate split on plane of spiral. 79½ mins.—water again applied. 80¼

mins.—pieces fell on west, exposing spiral at 6½ ft. up. 80¼ mins.—pieces fell on west, exposing spiral in upper section. 82 mins.—one-half of the circumference of the spiral exposed on west up to 9 ft. above base and one-fourth of its circumference exposed above this point; little change during past two minutes except concrete washed out between wires. 84¼ mins.—water shut off; total duration of water test, 5 mins.

After Test.—Concrete washed off to spiral all around in middle section, 4 to 8 ft. up. In lower section spiral exposed all around 1 to 4 ft. up, except for strip about 12 in. wide on east. In upper section, spiral exposed for about half of circumference on west, 8 to 10½ ft. up and for about one-fourth of circumference up to 12 ft. above base. Column expanded 0.14 in. during the fire test and contracted 0.045 in. during the water test.

Load Test.—On the following day the column was loaded until failure occurred at 536 000 lbs., 6½ ft. above base; all bars buckled out 1¼ in., and the spiral wire broke in tension with reduced section; faint cracking sounds were first heard at about 300 000 lbs.; at 400 000 lbs. the compressive deformation due to a given load increment was about twice that at the start, and at 500 000 lbs. about five times the initial rate. (Figs. 89 and 145.)

(*f*) UNPROTECTED CAST-IRON COLUMNS

Test No. 114. Round Cast Iron. Vertically cast; unprotected

Fire Test.—22½ mins.—gas shut off; column glowing low red in lower half; no visible deflection.

Water Test.—23 mins.—water applied to column; 30-lb. pressure. 23¼ mins.—column still glowing on north, east, and south. 23¾ mins.—still glowing on east. 24 mins.—water off; duration of water test 1 min.; column still glowing in lower third on east.

After Test.—Column unaffected as far as could be seen except for deflection of 1⅜ in. to west, 4 ft. above base. Column expanded 0.99 in. during the fire test and contracted 0.51 in. during the water test. (See Fig. 46.)

Load Test.—On the following day the column was loaded to failure and supported a maximum load of 527 000 lbs.; pumping was continued, but load fell off gradually until at 428 000 lbs. column broke in two with great violence 6 ft. above base; during the loading the column deflected ⅛ in. to west at center at 445 000 lbs., and ½ in. at 520 000 lbs. fracture coarse, gray, crystallline; break very jagged. (Figs. 89, 146, and 46.)

Test No. 115. Round Cast Iron. Vertically cast; unprotected

Fire Test.—20 mins.—trace of color visible on column. 30 mins.—gas shut off; column glowing low red entire length; no visible deflection.

Water Test.—31¼ mins.—water applied to column; 30-lb. pressure. 31½ mins.—column still glowing on north, east, and south. 31¾ mins.—still glowing on east. 32 mins.—still glowing in lower half on east. 32¼ mins.—water off; duration of water test, 1 min.; no color on column.

After Test.—Column unaffected as far as could be seen except for deflection of ⅞ in. to west, 5½ ft. above base; column expanded ⅞ in. during the fire test and contracted ⅝ in. during the water test. (See Fig. 46.)

Load Test.—On the following day the column was again loaded and supported a maximum load of 502 000 lbs.; pumping was continued with load decreasing, until at 348 000 lbs. it was removed to avoid breaking the column; during the test, column deflected ¾ in. to east at center at 290 000 lbs., when deflection changed direction; at 450 000 lbs. it was ⅛ in. east, increasing rapidly to the west, until at maximum load it was 1¼ in. west; deflection continued to increase until load was removed, when column recovered a large part of the deflection. (Figs. 89, 145, and 46.)

XII. GENERAL SUMMARY AND DISCUSSION

Included under this head are considerations relative to the quality of the materials of the test columns and their coverings; the general effects of load, fire, and water; the useful limit of the various types of columns as loaded and exposed to fire; test durations, effects of methods of application, and causes of variation in results.

1. CHARACTERISTICS OF COLUMNS AND THEIR MATERIALS

These relate to the structural quality of the test columns and the physical and thermal effects of load and fire.

(*a*) STRUCTURAL-STEEL COLUMNS

(1) MATERIAL AND FABRICATION.—The structural steel for the test columns was made by the open-hearth process in four mills in different localities, and in point of chemical and mechanical properties came fairly within accepted specification limits. Measurements of area of structural sections indicated general agreement with the nominal or handbook areas within 1 per cent, although a few measured areas differed from the nominal by amounts up to 4 per cent.

The columns were detailed according to standards of current practice, and as furnished fabricated by five companies no defects due to faulty design or fabrication were noted before test or developed as a result of the tests, except that the bearings as received were too uneven to insure uniform distribution of load.

A number of structural-steel columns that had been subjected to fire and water tests in the protected condition without injury to the steel were subsequently stripped of their covering and loaded to failure, the maximum sustained being 30 600 to 37 600 pounds per square inch, which represent ultimate factors of safety from a little less than 3 to over 4 on the computed working loads.

(2) EFFECT OF SLENDERNESS RATIO.—The slenderness ratio $\left(\frac{l}{r}\right)$ within the limits 40 to 80 appears to have little influence on the strength of structural-steel building columns as exposed to fire. In the case of some of the columns of higher slenderness ratio deflections were larger at a given stage of the test than for columns of more rigid section. The columns had flat end bear-

ings, the upper end being fully restrained and the lower end partly restrained by the base angles and anchor bolts.

(3) LATERAL DEFLECTION.—The direction of the lateral deflection before and after failure conformed with the line of least rigidity of the section, except in a few cases where the direction of the deflection was influenced by local heating of portions of the column. Deflections immediately before failure of a little over 2 inches were noted in a few tests, although generally they were between 1½ and 2 inches. Decided lateral deflection did not, as a rule, develop until the point of maximum expansion had been passed.

(4) VERTICAL DEFORMATION.—Steel columns when loaded and exposed to fire expand up to a point where the rate of compression of the metal due to the load becomes equal to or greater than the thermal expansion. The total expansion varied from ⅞ of an inch for the columns nearly uniformly heated over the full exposed length to ⅛ of an inch for those subjected to local heating due to failure of portions of the covering.

The measured expansion per unit length up to the point of maximum expansion, as measured over a 37-inch gage length, varied from 0.0044 to 0.0066, the lower values being due mainly to local heating. The unit expansion per °C. rise of temperature, or coefficient of expansion of the columns as loaded and exposed to fire, averaged 0.0000125 (0.000007 per °F.) as taken from room temperature up to the point where decided yielding of the metal was apparent.

(5) LOAD CARRIED BY THE COVERING.—On application of working load, the covering takes portions of the load proportionate to its area and rigidity with reference to the steel. With increase of temperature the higher rate of expansion of the steel causes a larger portion of the load to be transferred to the steel section, this condition continuing up to the point of maximum expansion. Subsequently the compressive yielding of the steel causes load to be again transferred to the covering, the amount depending on the stability and load-carrying capacity retained by the covering materials after the fire exposure.

(6) AVERAGE EFFECTIVE TEMPERATURES.—For the steel columns for which average effective temperature determinations were made (Table 43), the range at maximum expansion was

from 484 to 593° C (903 to 1099° F), with an average of 530° C (986° F). At failure the average temperature range was from 570 to 837° C (1058 to 1539° F), the average of all determinations being 668° C (1234° F). The applied working loads if carried by the steel alone correspond to stresses of 8900 to 14 500 pounds per square inch, as varying for the different structural sections, the average being 11 600 pounds per square inch (Table 41, p. 97).

The lower temperatures given above correspond nearly with those that hold for steel at maximum expansion and at failure under the given unit loads. The higher temperatures obtained in the tests where the covering carried portions of the load, this effect being much less marked at maximum expansion than at failure.

(7) General Cause of Failure.—The failure in the fire test was due in all cases to decrease in mechanical strength of steel with increase of temperature. The temperature required to cause failure depended mainly on the unit load carried by the structural section, although uneven stress distribution as caused by incidental eccentricity of load application, uneven bearings, and deflection of the column, entered as possible modifying conditions. The general or local lateral deflections occurring immediately before failure were due to yielding of the metal and can be considered as failure effects.

The deflection and distortion at failure caused large permanent loss of load-carrying capacity, depending on the amount of the deflection and the rigidity of the section, the remaining strength being estimated at 5 to 50 per cent of that before test.

(*b*) CAST-IRON COLUMNS

(1) Material and Manufacture.—Of the 10 cast-iron columns tested, 7 were cast horizontally and 3 were cast in vertical position. The iron of the horizontally cast columns conformed with published specifications for gray-iron castings in point of chemical and physical properties, the specimens being cut from the ribs in the head section of the columns. The tests made on the iron of the vertically cast columns were too few to be conclusive, but they indicated difference in the strength of the iron of 15 to 20 per cent as between the two ends of the column (Table 9, p. 339).

The wall thickness of the columns differed from the nominal thickness by a maximum of ¼ of an inch for the horizontally

cast columns and $\frac{1}{16}$ of an inch for the vertically cast pipe columns, the difference being caused mainly by displacements of the molding core in casting the columns.

Two of the vertically cast columns that had been subjected to fire and water tests, which induced permanent lateral deflections in one of $\frac{7}{8}$ inch and in the other of $1\frac{3}{8}$ inch, were subsequently loaded to failure, the maximum sustained being 36 400 and 34 700 pounds per square inch, respectively.

(2) DEFORMATION AND TEMPERATURE.—The same general characteristics of deformation and temperature obtained for the cast-iron as for the steel columns. The average unit expansion attained by the cast-iron columns was 0.0064 against 0.0054 for the steel columns, the average temperature at maximum expansion and at failure being about 70° C (126° F) higher than as obtained with the steel columns, the difference being due to the lower allowable working loads applied to the cast iron. The average applied unit loads, if assumed carried by the metal section alone was 6500 pounds per square inch as compared with 11 600 pounds per square inch average for the steel columns, a difference that was considerably reduced by the interaction of the coverings, which were heavier around the steel columns and generally carried larger proportions of the applied loads than those of the cast-iron columns.

(3) CAUSE AND CHARACTER OF FAILURE.—Failure in the fire test was primarily due to inability of the metal to sustain load at the given temperature, the bucking and fracture incident with failure being in the nature of failure effects. The partial or full fracture of the metal section at one or more points, resulted in almost complete loss of load-sustaining capacity. The direction of the deflection for the columns having uneven wall thickness was quite uniformly toward the side with the heavier thickness, due evidently to compressive yielding of the thinner and more highly stressed metal on the opposite side.

(c) PIPE COLUMNS

(1) MATERIAL AND MANUFACTURE.—No tests of the metal of the pipes were obtained, but the material was apparently mild steel. The metal was of standard thickness. Cylinders taken of the 1:1½:3 concrete mixed for the filling of the plain pipe

column developed an average compressive strength of very nearly 4000 pounds per square inch at the time the column was tested.

The columns were filled at the manufacturer's plant and furnished complete with bearing details, the latter being arranged to approximate conditions of use in buildings (Fig. 7, p. 24).

(2) DEFORMATION AND TEMPERATURE.—The metal being exposed, attained early in the fire test a higher temperature than the filling, and expanding away from the latter, would assume most of the applied load, the unit stress, if all of the load was carried by the pipe metal, being 16 500 pounds per square inch for the plain concrete-filled pipe column. The point of maximum expansion was in this case well defined and was attained at an earlier period and lower temperature than in tests of steel and cast-iron columns of about the same duration, due to the higher load sustained by the metal. The compressive deformation subsequent to maximum expansion was large and developed the strength of the concrete filling before failure occurred.

The reinforced-pipe column which had structural angles embedded in the concrete filling, expanded only a small amount during the fire test, yielding of the pipe metal under the high induced stresses beginning early in the test. Toward the end of the test the temperature of the pipe was so high that it could have sustained only a small part of the load, which must have been carried mainly by the steel reinforcement with stresses too high to permit much expansion.

Lateral deflections began at about the time the pipe metal began to yield and increased as failure was approached.

(*d*) REINFORCED-CONCRETE COLUMNS

(1) MECHANICAL PROPERTIES OF THE CONCRETE.—Made under conditions approximating those obtaining in building construction, the concrete developed wide variability in strength and elastic properties, the principal cause of which was difference in water content of the concrete mixture. The average compressive strength of 8 by 16 inch cylinders of 1:2:4 limestone and of trap-rock concrete made from the concrete mixed for the reinforced-concrete columns was 1525 pounds per square inch at 28 days and 1930 pounds per square inch at 16 months, with maximum variations above and below the averages of 67 and 54 per

cent, respectively. The modulus of elasticity at 650 pounds per square inch had an average value of 3 080 000 pounds per square inch with concrete aged 16 months, its variability being somewhat greater than that of the compressive strength. This variability in strength and elastic properties appears to have had little influence on the fire resistance of the concrete.

(2) DEFORMATION AND TEMPERATURE.—Maximum unit expansion of 0.0023 to 0.0046 were observed in fire tests of reinforced-concrete columns, the average being about one-half of that found for cast-iron columns. The indications are that a considerable portion of the expansion was due to the vertical steel reinforcement, maximum expansion being coincident with temperatures in the reinforcement of 400 to 500° C, in which region yielding of the metal takes place under stresses that were likely to be induced in the bars by their higher expansion rate with reference to the concrete. With limestone or trap-rock concrete and with lateral ties not less than 12 inches apart no tendency for the vertical bars to buckle before failure was noted.

Following maximum expansion, the columns gradually compressed, the rate being less rapid than for steel and cast-iron columns (Figs. 169 to 171). Temperatures at failure in the center of the trap-rock concrete columns that failed in the fire test averaged 450° C (842° F), and in the vertical reinforcing bars 894° C (1641° F).

(3) CHARACTER OF FAILURE.—The columns failed locally by compression or by combined compression and shearing on inclined planes, the vertical bars buckling and lateral ties or hooping, breaking, or yielding at the point of failure. No large lateral deflections developed either before or at failure.

(*e*) TIMBER COLUMNS

(1) QUALITY OF MATERIAL.—The timber was long-leaf pine and Douglas fir of select structural grade and conformed with the requirements of published specifications for structural timber (Table 2).

(2) DEFORMATION AND TEMPERATURE.—Except for slight expansions noted during the first few minutes of the test, the timber contracted or compressed under the combined load and fire condition, most of the deformation occurring in the wood at its bearings on the metal cap introduced near the top of the column, the crushing of the wood at these points causing progressive depression of the top of the timber columns (Fig. 47, p. 126).

The temperature in the timber away from the surface and the cap bearings was retarded by the evaporation of moisture and the low heat conductivity of the wood and did not exceed 100° C (212° F) until near failure (Figs. 140 and 141). In the metal cap at the edge of the column bearing the temperatures at failure varied for the different tests from 432 to 544° C (810 to 1011° F) (Table 43, p. 122).

(3) CAUSE OF FAILURE.—The cause of failure of the timber columns was loss of strength of the wood at the cap bearings, due to conduction of heat from the flanges of the metal caps to the bearing plates and into the wood. The consequent softening of the wood caused the columns to slip laterally on their bearings with the steel-plate caps and to fracture the cast-iron caps. While the test fire reduced the area of the columns by 29 to 55 per cent their resistance to fire and load outside of the bearings was not fully developed in the tests.

2. USEFUL LIMITS

The useful limit point with reference to fire exposure of columns is here taken as the limit of the period within which the effects on the elements designed to carry load do not permanently impair their essential structural properties to such extent as to make them unsuitable for further use.

(a) STRUCTURAL-STEEL AND CAST-IRON COLUMNS

The test characteristics indicate that for columns restrained at the ends and with slenderness ratio not exceeding 80, the useful limit is approximately equal to the period of expansion as given in Table 43 (p. 122) and Fig. 45. The measured center deflections of columns within the given $\frac{l}{r}$ limit did not exceed ½ inch at the end of the expansion period. The deflection of some of the columns of higher slenderness ratio was larger, although at the given stage all measured deflections were within 1 inch. Since failure in details such as riveting was not apparent in any test, unfavorable conditions in this respect could not have existed at any time preceding failure.

Recent tests made by the Bureau of Standards on specimens of structural shapes that were heated under load up to the point of maximum expansion and cooled indicate little or no distortion of shape, loss in strength, or change in elastic properties. The compressive set on cooling and removal of load was not large enough to be objectionable as applied to a building column.

In the case of the structural-steel columns with 2-inch Chicago limestone or Joliet gravel concrete coverings the interval between maximum expansion and failure was equal to 73 per cent, or more, of the expansion period, and, considering the consequent lower rate of deformation of the column, the useful limit may possibly extend considerably beyond the expansion period. The same applies to the columns with 4-inch concrete coverings made with the Chicago limestone aggregate, which withstood the fire test in excess of 8 hours.

Damage to the covering is not considered in the above discussion, as it is commonly not regarded as a load-carrying element.

(*b*) PIPE COLUMNS

For the unreinforced pipe column the useful limit can be taken as the period of expansion in the fire test.

For the reinforced pipe column the transfer of load, as caused by successive heating and expansion, first to the pipe and later to the structural steel reinforcement, confounds the relation existing between maximum expansion and useful limit. The temperatures on the outside of the pipe indicate that the useful limit was reached after between 20 and 30 minutes of fire exposure. The center lateral deflection at the end of this interval was ¾ of an inch.

(*c*) REINFORCED-CONCRETE COLUMNS

The reinforced-concrete columns expanded less than the steel and cast-iron columns, reached their point of maximum expansion at a relatively earlier stage, and the subsequent compressive deformations and center deflections were smaller. Comparisons of the temperatures of the reinforcing bars and the attending unit deformation indicate the presence of high compressive stresses, although with concrete of the given aggregates and the methods of tying employed no tendency to buckle before failure was noted. For the columns that withstood the 8-hour fire test and sustained large additional loads before failure it is believed that the useful limit extended up to the end of the fire test. In the case of the columns that failed in the fire test or sustained but small additional load after 8 hours the matter is more indeterminate, although it can be stated that large compressive deformations did not obtain until within the 1½ hours preceding failure.

These considerations do not preclude damage to the outer concrete covering which it might be necessary to repair or replace after fire exposures within the useful limit.

(*d*) TIMBER COLUMNS

The effect that limits the usefulness of timber columns after fire exposure, as far as it concerns the timber, is surface damage from burning away of the wood, with consequent reduction of effective section. Large compressive deformation of the column due to impairment of the unburned wood from the heat apparently obtains only for points nearer failure. The amount of reduction in area that can be allowed depends on the basis of design. In common with general practice, the whole area of the test columns was assumed to carry load, no portion being deducted as protection. Assuming that the initial factor of safety was no more than adequate, minor reductions in area would render the column unfit for further use. Practical requirements relative to minimum and standard sizes often result in loadings lower than the safe working limit, in which case proportionate reductions in area due to fire could take place without impairing the safe load-carrying capacity.

With reference to fire effects on unprotected steel and cast-iron bearings, the rapid rates of deformation obtaining after 20 minutes in tests of unprotected timber columns indicate that the useful limit in this particular did not extend much beyond this period (Fig. 47, p. 126). For the protected timber columns the useful limit in this respect extended to within about the same time interval (15 to 30 minutes) before failure, as in the tests of unprotected timber columns.

(*e*) PRACTICAL APPLICATION

The above conclusions relative to useful limits find application where it is desired to design and protect columns so they will not be materially injured by a fire that consumes the contents of the structure supported by them. In applying the results of these tests for this purpose it is necessary to make deduction for variations in material and workmanship not developed in the tests. The percentage reduction can be safely taken as that used in deducing ultimate fire resistance periods in the concluding section of this paper. The period of expansion is influenced to less extent by incidental conditions, such as load-carrying capacity of covering materials, than is the time to failure.

While the useful limit may be made the basis of design of columns and coverings in special cases, it is believed that the most general use of the test results will be that based on the ultimate fire-resisting point; hence no detailed application of the former will here be made.

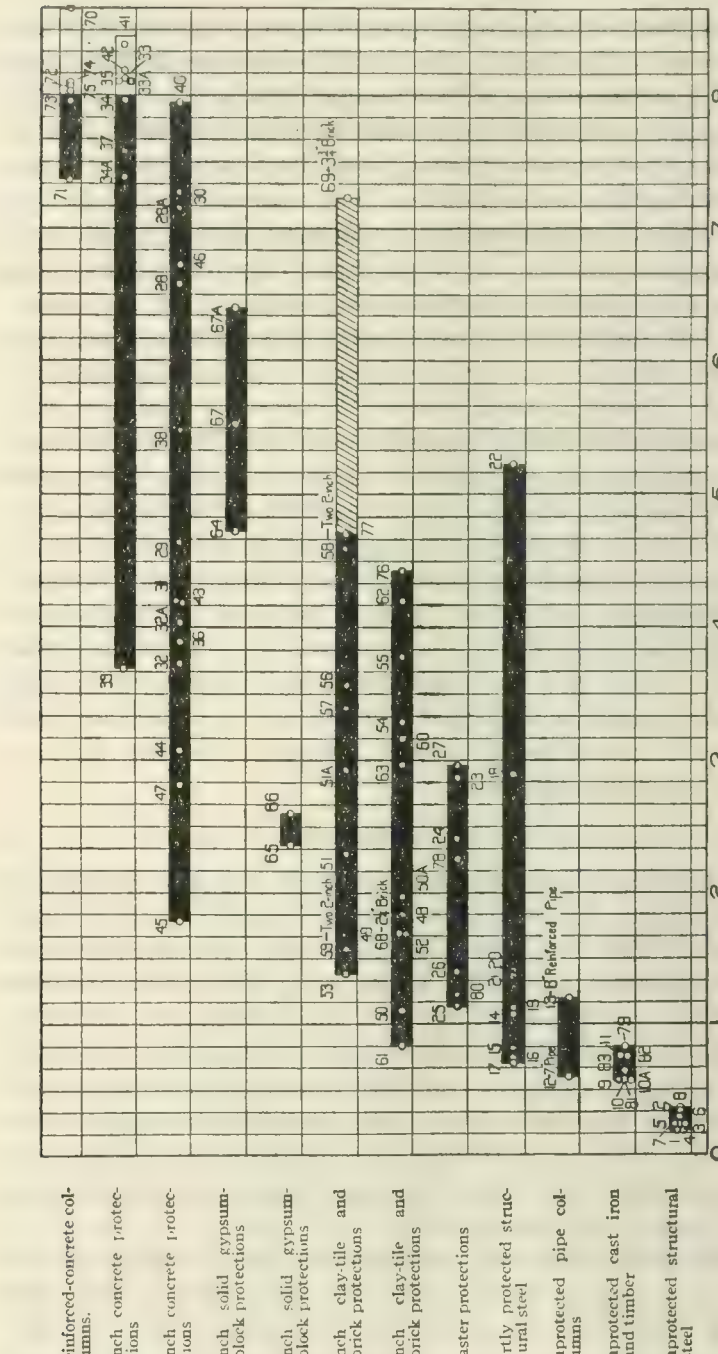


FIG. 48.—*Results of fire tests compared by groups*

3. DISCUSSION OF TEST DATA

The considerations herewith given relate to test duration, period of expansion, and the causes immediately affecting them. Comparisons of the time to failure of columns in the fire-test series arranged by groups are given in Fig. 48, which also provides a measure of the variation obtaining within each group. Further comparisons of time to failure and period of expansion are given in Table 43 (p. 122) and Figs. 42 (p. 114) and 45 (p. 121). In the discussion the time periods will be given to the nearest minute.

(*a*) DIFFERENCE IN COLUMNS, TEST EXPOSURE, AND SERVICE CONDITIONS

(1) VARIATIONS DUE TO DIFFERENCE IN COLUMNS.—Among the effects tending to produce variation in results with given types of protection for the same thickness of covering are size and shape of structural section and consequent differences in intensity of loading and resistance to fire; differences in fire-resisting properties of subclasses of covering material within each group, such as concrete of the different aggregate combinations and hollow clay tile of the various types of clay; variations in methods and details of application; incidental variations in material and workmanship. These are discussed in connection with each group whenever the test results developed information on any given particular.

A fairly wide range of structural section was employed in the tests, and it is believed that the influence of this factor was sufficiently determined.

One or more representative materials from each of the main subclasses of covering material were introduced. It is appreciated that considerable differences in mineral composition and structure may exist between material in a given subclass as occurring in different localities, and, while the difference in fire-resisting properties may be of minor order, wide application of the results of the tests will have to be made with care until the knowledge on the subject has been extended by further tests.

The full range in results due to difference in methods of application and in incidental variations in materials and workmanship was not developed in the tests, as this required a greater number of test duplications than could be introduced. In general, poured coverings are subject to smaller incidental differences than those built

up from small units; also, if stability is assured, a large element of variability is eliminated. It is also thought probable that columns and protections in buildings are subject to greater variability than those constructed for the tests, the latter representing workmanship of more uniform quality.

(2) VARIATIONS DUE TO DIFFERENCE IN LOAD AND FIRE CONDITIONS.—In the tests the amount of load applied on a given column section or column type was subject to minor variations only, the errors being generally within 1 per cent. Comparisons of the initial deformations produced in opposite flanges of individual columns indicate maximum outer-fiber stresses in several tests up to 25 per cent above what would obtain for uniform distribution of load. These can be ascribed mainly to bending stresses induced by uneven bearings and are no larger than those generally incident with compression tests or with normal loading of columns in buildings.

The loads applied in the tests were about 10 per cent higher than those derived from the column formula employed. While floors in buildings are subject to overload by more than this amount, the resultant load on the columns is likely to be reduced by the distribution of load concentrations to several columns and by the equalizing effect of loadings from several stories.

As exposed to fire in buildings, individual columns may take added load due to higher temperature, and consequent greater expansion, than the adjacent columns of the story, particularly if they are not fully protected. For steel or cast-iron columns the additional load assumed is limited by the ultimate flexural strength of their connections to the floor beams framing into them. In the case of interior columns the additional load possible to assume on this basis will seldom exceed 20 per cent of the nominal floor loads. Reinforced-concrete columns have more rigid floor connections than steel or cast-iron columns, but, due to their lower expansion and rate of temperature rise, more unfavorable loading conditions as due to unequal expansion are not likely to occur.

The furnace exposures are given in the test results as percentages of the average of all tests, the quantity compared being the area under the average time-temperature curve of each test. For the shorter tests with upper limit of duration of about one hour the difference in exposure may have had a considerable influence on results, considering also that the percentages given do not indicate

the full measure of the difference due to lag of the pyrometers. For the longer tests, variations in test results due to difference in furnace exposure were apparently of minor importance.

Comparing the average time-temperature curve of the column tests with the reference curve (Fig. 39) a fair agreement is seen to have been attained. The reference curve was adopted as being consistent with furnace exposures used in previous fire tests, and there was at the time no distinct understanding whether it should represent indicated temperatures or temperatures corrected for lag and radiation effects, little information on the extent of these effects being available. Investigations conducted subsequent to the completion of the column tests disclosed large effects due to lag during the initial period of the fire exposure and smaller, but more persistent, effects due to radiation (p. 101–106, Figs. 40 and 41). Any time temperature relation adopted as standard is necessarily more or less arbitrary, particularly for the initial period. Temperatures attained in fires afford some guidance, although they indicate a wide range in intensity. From the information available, points on the reference curve, considered either as indicated or as corrected temperatures, while not the maximum attained in fires of exceptional intensity, represent severe fire conditions.

(*b*) UNPROTECTED COLUMNS

Included under this head are structural steel, cast-iron, and pipe columns that were tested without protective coverings, all parts of their sections being assumed to carry proportionate portions of the applied load.

(1) STRUCTURAL-STEEL.—The time to failure of the unprotected structural-steel columns varied from 11 to 21 minutes (Table 42*a*, p. 108). The difference in results for the various column types, while due in part to variation in furnace exposure, is attributable also to difference in thickness of metal and in the unit loads sustained, sections with thin members under the higher unit loads failing sooner than sections whose members were arranged to form heavy metal thickness. Fig. 49 is a plot between unit load sustained, and time to failure in tests of unprotected structural-steel columns, showing the inverse relation obtaining between them. The variation in the unit load applied was due to difference in slenderness ratio between the different sections.

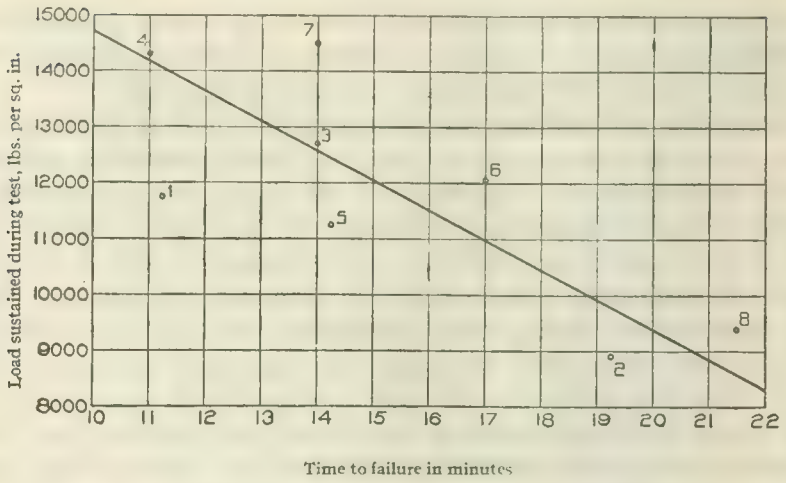


FIG. 49.—*Effect of load on fire resistance, unprotected structural-steel columns*

The average time to failure of the eight structural-steel sections was 15 minutes and the average period of expansion was 13 minutes (Table 43). Maximum temperatures from 578 to 668° C (1072 to 1234° F) were attained on the outside of the metal near the edges. It is difficult to estimate the average effective temperature, since the rise was too rapid to permit assuming temperature uniformity over the thickness of the metal. Tests recently made by the Bureau of Standards on small specimens indicate that for the loads sustained by the unprotected columns the failure temperatures of the structural steel fell within the limits 550 to 650 ° C (1022 to 1202° F).

(2) CAST-IRON.—The average time to failure of the unprotected and unfilled cast-iron columns, one of which was tested with ends restrained and two with unrestrained ends, was 34 minutes, the variation being less than one minute for the three tests. The periods of expansion varied from 22 to 24 minutes. The longer test periods and higher temperatures attained as compared with the structural-steel columns can be attributed largely to the lower allowable unit loads applied to cast iron and the relatively smaller surface exposed to the fire. As judged by the direction of the lateral deflection, failure in compression appears to have started on the side having the thinnest metal. Filling the interior with concrete (test No. 11) increased the time to failure 11 minutes, with a smaller proportionate increase in the length of the expansion period.

In the two fire and water tests of cast-iron columns, water was applied when the columns had attained maximum expansion, the metal being at low red heat. No cracks developed in the metal, the only effect being permanent lateral deflections of, respectively, 3/8 and 7/8 inch toward the side on which water was applied.

(3) PIPE COLUMNS.—The time to failure of the 7-inch pipe column was 36 minutes and maximum expansion was attained at 14 minutes.

The 8-inch reinforced pipe column failed after 1 hour 12 minutes, and maximum expansion was attained at 52 minutes. The column expanded only a small amount after 25 minutes, the total expansion being about 1/4 of an inch. The high temperatures on the surface of the pipe indicate that during the last half of the test period nost of the load was carried by the structural-steel reinforcement.

(c) PARTLY PROTECTED COLUMNS

Filling the reentrant portions or interior of the structural-steel sections with concrete greatly increased their fire-resisting property as compared with that in the unprotected condition (Table 42*b*). The difference was due to slower temperature rise in the metal resulting from the heat insulating and absorbing properties of the filling and to the load-carrying capacity of the concrete, the longer time intervals between maximum expansion and failure being due in great part to the latter factor.

Temperature differences up to 450° C (810° F) existed between the exposed flanges and the protected webs, the temperature of the latter at failure being generally below 500° C (932° F).

(1) EFFECT OF SECTION AND SIZE.—No decided effect due to variation in shape of structural section was noted except as it affected the size of the column.

In Fig. 50 is shown the relation between time to failure and area of material in cross section for all tests of partly protected columns, except No. 22, which was eliminated as being in effect a protected column. The relation shown accounts for the variation in results within the group as being due in large part to difference in size.

(2) EFFECT OF CONCRETE AGGREGATE AND TIES.—In the tests of short duration the concrete aggregate appears to have affected

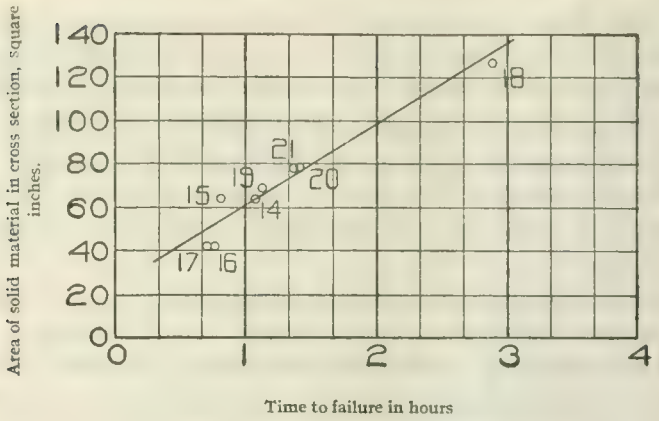


FIG. 50.—*Effect of size, partly protected columns*

results to a minor extent only, while in those of longer duration (Nos. 18 and 22) the results compare with those obtained for concrete protections, which they more nearly resemble.

Metal ties were placed in the concrete of all partly protected columns where not contained by the section members as in the latticed columns. No tendency was noted on the part of the filling to spall, buckle, or otherwise come loose before failure. The extent to which the ties functioned to prevent such effects is not fully determinate, although in tests of concrete protections made with the same aggregate little cracking or spalling was noted within the given time periods.

(*d*) PLASTER ON METAL-LATH PROTECTIONS

(1) MATERIAL AND DESIGN.—The Portland-cement plaster, of proportion $1 : \frac{1}{10} : 2\frac{1}{2}$ volume parts of Portland cement, hydrated lime, and coarse lake sand, as mixed by the workmen for the protections and tested in 2-inch cubes, developed an average compressive strength of 1677 pounds per square inch at 28 days and 2623 pounds per square inch at an average age of 16½ months, with maximum range in individual tests of 44 per cent below and 60 per cent above the given averages (Table 25, Fig. 23). Cubes made in the laboratory of the same materials and average water content (16.7 per cent) gave lower compressive strength, both as stored in air and in water (Table 28, p. 356).

The metal lath was wrapped around the structural-steel columns on bars or on pressed-steel channels acting as spacers, and the plaster applied in layers ⅞ to 1⅛ inches thick, each consisting of two body coats. The double-layer protections had a ¾-inch air space between layers. For the cast-iron column the plaster was applied on high-ribbed metal lath supported directly on the column. A broken air space of about ½ an inch thickness was formed next to the metal, due to failure of the plaster to fully fill the space back of the lath.

(2) Test Results.—The average time to failure of the structural-steel columns with single layer protection was 1 hour 16 minutes, and of those with double-layer protection 2 hours 38 minutes, the two tests of each varying from the average by less than 15 minutes (Table 42*c*, p. 109). The cast-iron column protected by a single layer of 1½-inch average thickness with a broken air space between it and the iron (test No. 27) stood up longer by a few minutes than any of the other columns in the group.

The temperature distribution across the section was generally very uniform, and the variations in the length of the column were not large (Figs. 97 and 98). This was due to the air space between the covering and the structural section, which permitted free heat interchange in the column, unmodified by the temperature gradients in the covering material.

(3) Cracking Due to Expansion of Covering.—During the first 20-minute period in all tests of Portland-cement plaster protections, cracking and disruption of the plaster took place below the bracket near the top of the column. This was evidently due to expansion of the plaster layer, which was restrained at the top of the column and at the bottom bearing. This effect appears to have had little influence on the time to failure in the given tests, the region of maximum column temperature and failure being in all cases within the middle 4 feet of the column height.

The sand used in the plaster was high in insolubles (chiefly silica) and low in calcite and dolomite (Table 14, p. 342).

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(4) Effect of Variation in Details of Application.—Comparing the layer thickness of test No. 23, where the plaster was applied on expanded metal lath, with that of test No. 24, where woven-wire lath was used, and also the outer layer thickness with the inner in test No. 110, applied, respectively, on expanded metal and on woven wire, a heavier layer thickness by ⅛ inch was attained in all cases with the expanded metal (Tables 3*d* and 4*d*). This difference in layer thickness may account in part for the longer test duration of No. 23 as compared with No. 24. Further indications that layer thickness is an important element in the protection given is had in the case of tests Nos. 25 and 26, where, with a difference in layer thickness of ⅛ inch, a difference in time to failure of 16 minutes obtains.

The tests developed no evidence that the method of supporting the lath had any influence on results (cf. Sec. IV, par. 2*a*, p. 56).

All double coverings had an air space about ¾ of an inch wide between the inner and outer layers. No tests were made with a single layer equivalent in thickness to that of two double layers, hence no direct evidence relative to the value of the air space as an insulating medium was obtained.

(5) Effect of Water Application.—The water carried away some loose pieces near the top of the column where cracking and spalling had taken place during the fire period (Fig. 87). It exposed the lath at the corners of the outer layer for portions of the height. The test did not appear to have materially injured the insulating value of the covering since the time to failure in the subsequent fire test nearly equaled that of the corresponding test (No. 23) in the fire series (Table 44, test No. 110).

(*e*) CONCRETE PROTECTIONS

(1) Mechanical Properties of the Concrete.—The compressive strength of 8 by 16 inch cylinders made from 1:2:4 gravel or crushed-stone concrete mixed for the column coverings under conditions approximating those of building practice, averaged 1520 pounds per square inch at 29 days and 2100 pounds per square inch at an average age of 15 months, the maximum range of individual test results being 118 per cent above and 57 per cent below the given average values (Table 21, p. 346). As shown in Fig. 15 (p. 66), the range in results increased with the number of tests in the group, indicating that in the groups with the smaller number of tests the full possibility of variation was not developed.

The principal cause of the variation in strength was apparently difference in consistency of the concrete mixtures. Increasing the time of mixing from one minute to two minutes gave an indicated increase in the average compressive strength of 45 per cent (Fig. 19, p. 69).

The modulus of elasticity of the concrete varied approximately with the compressive strength from less than 1 000 000 to over 4 000 000 pounds per square inch (Fig. 21). This large variability in the mechanical properties of the concrete appears to have had little influence on its fire-resistive properties, the latter depending chiefly on the mineral composition of the aggregates employed.

(2) Function of Concrete as a Covering Material.—Concrete applied as a protective covering or filling to steel or cast-iron columns retards the temperature rise in the metal when the column is exposed to fire and further retards the failure by carrying portions of the column load proportionate to its relative area and rigidity as compared with the metal.

The protections were applied as square or round coverings, generally 2 inches and 4 inches in thickness as measured from the surface of the covering to the metal. The time to failure in the fire tests varied from 1 hour 47 minutes to 7 hours 57 minutes for the 2-inch protections, and from 3 hours 41 minutes to over 8 hours for the 4-inch protections (Table 42*d*, p. 110).

(3) Variations due to Concrete Aggregate.—With a given thickness or size of covering the main cause of variation in results was the difference in fire-resisting properties of concrete made with different aggregates. In this particular the concrete can be placed in three groups. That giving the most unfavorable results was the concrete made with Meramec River sand and gravel, a number of large cracks forming early in the tests followed by spalling of large and small pieces of concrete not held by the ties (test Nos. 39 and 45). This sand and gravel consist almost wholly of quartz and chert grains and pebbles, the gravel having a particularly high chert content. Both minerals are forms of silica (SiO_2), the quartz being crystalline and anhydrous, and the chert amorphous, with a variable amount of water in chemical combination. On being heated part of the combined water in the chert is liberated, and the consequent vaporization disrupts the pebbles. Other causes of disruption of concrete made with

siliceous aggregates are abrupt volume changes, points of which are known to exist for chert as low as 210° C (410° F). Quartz has a decided point of abrupt volume change at 573° C (1063° F), where it is transformed into the mineral tridymite, the change extending over a considerable temperature range when the heating is rapid. Liquid inclusions contained in small cavities formed when the rock crystallized from the molten condition may be the cause of some of the cracking incident with fire exposure.

The middle group includes concrete made with trap rock, granite, sandstone, and hard-coal cinder.

In tests with trap-rock and cinder concrete a small amount of cracking developed during the last part of the fire period, but no spalling of note occurred before failure. In the granite-concrete protections the cracks developed earlier in the test, and portions of the corners spalled off during the last 30 minutes of the test period. In the tests with sandstone-concrete protections cracking and spalling of corners outside of the wire tie began in the first 30-minute period and continued during the next hour, after which there was little apparent change before failure. The spalling exposed portions of the flange edges, which to some extent hastened the failure. The average time to failure in tests with sandstone-concrete protections was intermediate between those with trap-rock and those with cinder concrete. The cracking of sandstone concrete after a short fire exposure can be ascribed mainly to the abrupt volume change of the constituent quartz grains as noted above.

Fusion of the trap-rock concrete occurred where the tests extended beyond 7 hours, the concrete being affected to a depth of about 1½ inches. Flowing of concrete due to fusion, while not general, occasionally formed pockets up to a 2-inch depth. Incipient fusion to about the same depth occurred in the 4-inch granite-concrete protections, although no actual flowing of concrete took place.

The third group comprises protections of Chicago-limestone concrete and Joilet-gravel concrete. The composition of this gravel is similar to that of the Chicago limestone, and the fire-resisting properties of the concrete made with each compare quite closely. Very little cracking resulted on exposure to fire and their heat-insulating value was increased by the change of the calcium and magnesium carbonate to the corresponding oxides.

This process retarded the flow of heat through the region of change and left material of good insulating properties. Immediately after test the surface of the concrete was firm, but after a few weeks exposure the hydration of the oxides caused slaking and crumbling of the calcined material (Fig. 62, p, 221).

(4) COMPARISON OF 2 AND 4 INCH PROTECTIONS.—In the comparison given in Fig. 51, the tests of concrete protections are arranged by groups as defined in the preceding paragraph, the line in the case of the middle group connecting the average value for each thickness. Test Nos. 40, 46, and 47 are omitted in this comparison on account of extreme shape and size of section, and No. 44 on account of leaner concrete mixture.

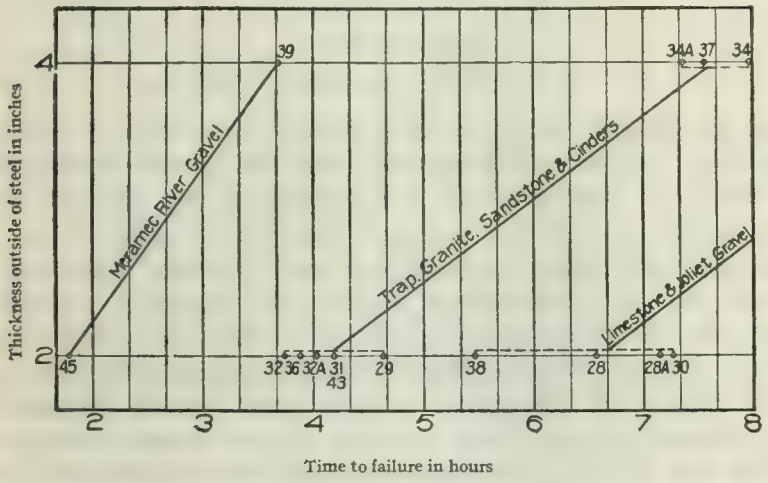


FIG. 51.—*Comparison of 2-inch and 4-inch concrete protections*

The time to failure under working load was not determined for the 4-inch limestone-concrete protections, as they were loaded to failure after withstanding the fire test in excess of 8 hours. They all attained maximum expansion within the last 30 minutes of the 8-hour fire period, and from comparison with results obtained with the corresponding 2-inch protections, failure in the case of the 4-inch protections would not have taken place before the end of 10 hours, assuming the same load and the same furnace temperature rise as obtained during the 8-hour period.

(5) Effect of Size.—In Fig. 52 the time to failure in tests with concrete protections of the middle group is plotted against the area of steel and concrete in the cross section. Variations

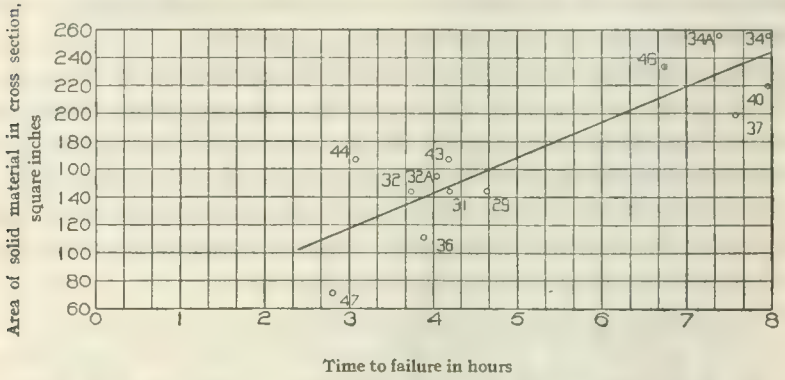


Fig. 52.—*Effect of size, concrete protections*

from the general trend can be accounted for as due to concrete aggregate, proportions of concrete mixture, type of section, and to incidental differences in test conditions and test columns. The extreme variation in furnace exposure as measured by the area under the furnace temperature curves is within 2 per cent as between all the tests plotted, except Nos. 34 and 34*A*, between which there is a difference of 4 per cent (Table 42*d*), which latter may be responsible for the difference in failure time of the two tests.

(6) Effect of Strength of Concrete.—Some decrease in fire resistance due to leaner mixture may be noted by comparison of Nos. 33 and 33*A* with No. 35, and No. 43 with No. 44.

No evidence was developed that variation in the strength of the concrete of the same aggregate and proportion of mixture had any appreciable influence on the results of fire tests of concrete protections. This was due to the large change in mechanical properties produced by the heat. Concrete as made with different aggregates preserves strength to different degrees on exposure to fire. This had a decided influence on results, the longer test periods, and particularly the longer intervals between maximum expansion and failure of the limestone concrete and Joilet-gravel concrete coverings, can be attributed in a great part to this cause.

(7) Influence of Shape of Section and Covering.—As in the case of the partly protected columns, the only well-defined effect of change in shape of structural section was primarily due to the resulting difference in the cross-sectional area of the covered column.

Round and square coverings of trap-rock concrete displayed only minor differences in the number and size of cracks that developed before failure. Fine vertical cracks 2 to 4 inches from the corners formed in the square coverings at somewhat earlier periods than the first cracks noted in the round coverings. In neither case do these cracks appear to have had any appreciable influence on the time to failure. Coverings made of concrete more subject to cracking may possibly develop greater differences due to shape, although with concrete made with highly siliceous aggregates, the influence of the aggregate is so large that other effects are small in comparison. This is shown in test No. 45, where the round, siliceous-gravel concrete covering sustained severe cracking and spalling early in the test which caused failure over one hour earlier than in any other test of concrete protection.

(8) Function of the Wire Tie.—In all tests of concrete protection, except Nos. 28A, 33A, and 47, the concrete was tied by a wire tie wound spirally around the structural-steel section. In Nos. 28A and 33A the concrete aggregate was Chicago limestone. No cracking of consequence developed before the end of these tests, and the absence of the tie had no influence on the results. In the case of the cinder-concrete protection in test No. 47 no cracking of note occurred until near failure and after the column had sustained large compressive deformation. About two minutes before failure most of the covering fell off. This would have been prevented if the wire tie had been present, although the temperature of the metal and the deformation of the column was such that failure was imminent.

It is not possible to state the extent to which the tie functioned in all tests of concrete protection, but it was undoubtedly of value where there was any tendency for the concrete to crack and fall off before failure. In the case of Meramec River gravel concrete and sandstone concrete, spalling of portions of the covering outside of the tie occurred early in the test, the concrete on the middle of the flanges and webs being apparently held by the ties.

(9) Effect of Water Application.—In the three tests where the wire was placed in the coverings the water pitted the exposed faces of the concrete and carried away portions of the corners and sides, leaving parts of the flanges and flange edges exposed. The damage was most marked in the regions where cracking was noted during the fire period (Figs. 83 and 84, p. 242–243).

In the case of the one covering that was not tied the water loosened or carried away most of the protection on the flanges, leaving the column in the condition of partial protection. Most of the damage was incurred after 2 minutes of water application, the total period being 5 minutes.

(*f*) HOLLOW CLAY-TILE PROTECTIONS

(1) Mechanical Properties of the Tile.—The average compressive strength of specimens of the hollow clay partition tile used in the column coverings was 5350 pounds per square inch as tested on end and 4370 pounds per square inch tested on edge, the maximum variation above these values being 134 per cent and below 78 per cent, of the lower average value. For the same type of clay the range in results was smaller. The strength was generally proportional to the density of the tile as indicated by percentage of porosity and of absorption (Table 31, p. 358–359).

In transverse tests of hollow tile the average computed outer fiber stress at failure was 527 pounds per square inch and the shear 233 pounds per square inch, the failure being apparently due to combined shear and tension. The range in results was larger than in the compression tests (Table 32).

There appears to be little relation between the mechanical strength and the fire-resistive properties of the tile, the latter depending mainly on the type of clay. This is significant as specifications based on the mechanical properties often disqualify tile desirable from the standpoint of resistance to fire.

(2) Test Results.—In tests of hollow-clay tile protections, using tile of the given types of clay applied according to the methods previously described, the time to failure ranged from 50 minutes to 4 hours 42 minutes. This large range in results was due to a number of factors that influence the effectiveness of this type of protection, including, besides type of clay, methods of manufacture of the tile, thickness of shells and webs, the presence or absence of concrete or other filling, and the methods used for tying the tile (Table 42*e*, p. 111).

(3) Variations Due to Type of Clay and Details of Manufacture.—The extent to which the tile cracked and spalled on exposure to fire varied with the type of clay of which it was made and the degree of hardness to which it was burned. All of the tile used in the coverings was straight, nonporous partition tile, burned without sawdust or other filling, except the round tile in test Nos. 62 and 63, which was porous.

The semifire-clay tile generally gave the most favorable results in the fire tests, and of the two represented the tile of medium hardness and having heavier webs and shells developed few cracks and little spalling before failure. In the coverings of round, porous semifire-clay tile a number of vertical cracks formed after a short fire exposure, which became wider as failure was approached. Few other disruptive effects were noted, almost all material remaining in place till the end of the test. Only minor differences in behavior were noted between tile made of the two kinds of surface clay, cracking and spalling of outer shells and buckling out of the tile being characteristic of tests of both. In the case of shale tile, these effects were even more pronounced, severe cracking taking place during the first few minutes of the test, followed by general spalling of outer shells.

(4) Comparison of 2 and 4 inch Protections.—In Fig. 53 is given a comparison of time to failure in tests with 2 and 4 inch

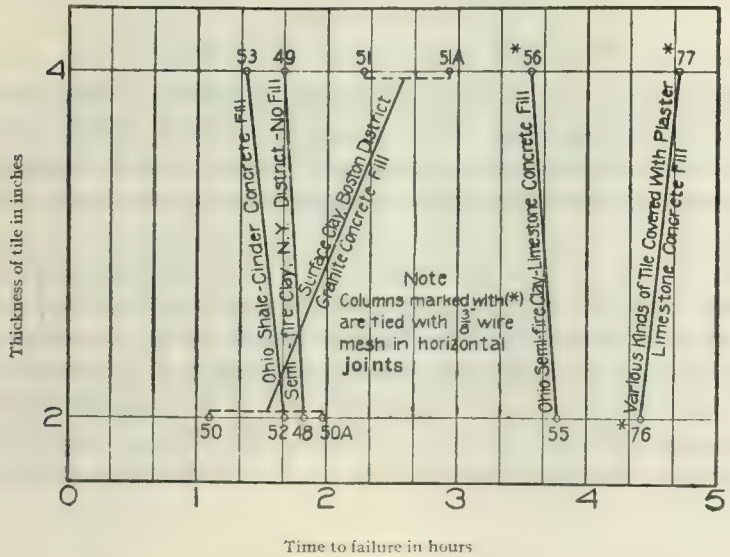


Fig. 53.—*Comparison of 2-inch and 4-inch hollow clay-tile protections*

hollow clay-tile protections, all other details being comparable except as noted. The tests show little difference between the two, the thickness of the air space and minor variations in thickness of shells having apparently little influence on results. The difference in results in test Nos. 50 and 50A as compared with 51 and 51A can be attributed to larger differences in thickness of shells and webs and also to the greater stability of the 4-inch tile set on end and tied with outside wire ties, as against the 2-inch tile laid flat in 6-inch courses without ties (p. 137-138).

(5) Effect of Size.—In Fig. 54 is shown the relation between the time to failure in all tests of hollow clay-tile protections made with nonporous partition tile and the total area of solid material

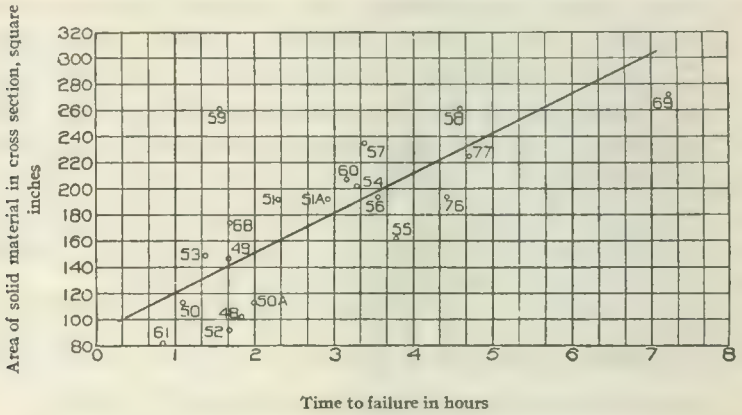


Fig. 54.—*Effect of size, clay-tile and brick protections*

in the cross section, a fairly consistent variation of the one with the other being evident. Nos. 58 and 59, having double layer of tile with tile filling, gave less favorable results in this comparison than tests of protections with a single layer of tile and concrete fill.

(6) Effect of Ties and Filling.—Fig. 55 gives a comparison of results attained with the two methods used for tying the tile and of including or omitting the concrete or tile filling. As shown on the diagram and confirmed by test characteristics, the mesh in the horizontal joints is a little more effective in holding the tile than the outside wire ties; also the concrete or tile filling is an important element in the protecting property of a hollow-tile protection.

The efficiency of wire mesh in the joints as against outside wire ties, in the case of protections of surface-clay tile with hollow tile filling, is shown by comparison of results in test Nos. 58 and 59. It should be considered in this connection that minor parts of the difference may be due to incidental variations in workmanship and test conditions, and also that protections of tile less subject to cracking and disruption on exposure to heat would show relatively smaller differences in results, due to the methods used for tying the tile.

The concrete filling not only serves as a protecting medium, but also assists in holding the tile in place by adhesion. In test

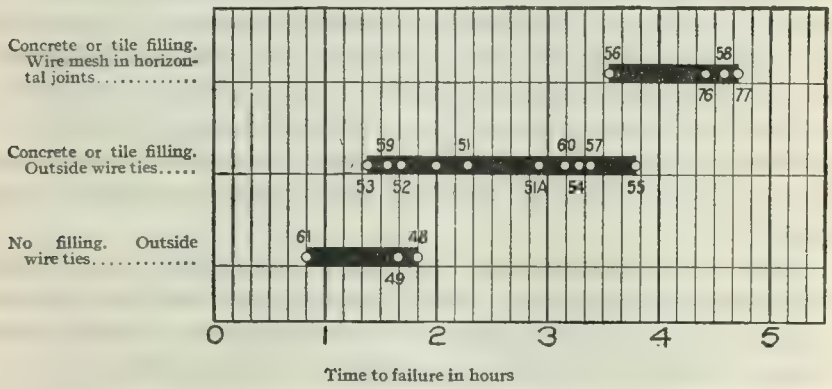


FIG. 55.—*Effect of ties and filling, hollow clay-tile protections*

No. 60 the concrete filling was placed before the tile was set, the tile being bonded to the filling with a thin mortar joint and tied with outside wire ties. A large number of tile units fell off early in the test, the behavior in this particular being distinctly different from that of protections with concrete fill placed after the tile was set, where up to points near failure the inner shell generally remained in place.

As indicated by the average temperatures in the steel at failure, the concrete filling carried portions of the applied load varying with the area, the higher temperatures at failure being generally

incident with the tests having the larger fills. In the latter tests the periods between maximum expansion and failure were comparable with those obtaining for concrete protections, whereas in the tests where the concrete fill was omitted or of small area this time interval was relatively short (Table 43, p. 122).

(7) Effectiveness of Plastering.—The tile in test Nos. 76 and 77 were covered with standard applications of gypsum and of lime plaster, respectively, applied 3 days after the concrete fill was placed, the columns being tested 42 and 45 days after plastering.

In test No. 76 the gypsum plaster began to fall off early in the test, exposing a few tile units at 2 minutes and more than half of the tile surface at 20 minutes. General cracking and spalling was probably delayed to some extent by the insulation given the tile by the plaster during the first few minutes of the fire exposure.

The lime plaster in test No. 77 fell off during the first half minute of the test, exposing about three-fourths of the total tile surface, its influence on the test result being apparently very small.

These results with plaster may not be applicable where it has seasoned for a longer time, and without further tests they should not be taken to hold rigidly for well-cured plaster coatings that from conditions of normal exposure are thoroughly dry.

(8) Effect of Water Application.—The water generally carried away the tile that had been decidedly damaged during the preceding fire exposure, although adjacent courses of relatively sound tile were in some instances carried down along with those impaired by the fire (Figs. 84 and 85). A large proportion of the effects took place during the first minute. In the test where the tile protection was filled with concrete the condition of the column after the water application approached that of partial concrete protection. In the case of the unfilled columns the steel in the region stripped of tile was unprotected, except for the portions of the mortar joint that adhered to the flanges.

No decided difference was noted between the outside wire ties and the wire mesh in the joints in effectiveness in holding the tile during the water application, although the comparisons were too few to afford definite conclusions.

(*g*) BRICK PROTECTIONS

(1) PROPERTIES OF THE BRICK.—The common brick used in the brick protections was a wire-end cut brick made in the Chicago (Ill.) district of calcareous surface clay. Compression tests gave average values of 3200, 1960, and 2815 pounds per square inch as tested on end, edge, and side, respectively, and average transverse strength of 862 pounds per square inch, with maximum variations above or below the averages of between 50 and 100 per cent (Tables 35 and 36). The brick was soft, with relatively low fusion point and high percentages of porosity and absorption (Tables 33 and 34, p. 361).

(2) TEST RESULTS.—In the two tests of columns protected by brick approximately the same relation obtained between time to failure and sectional area of the covered columns as for the hollow clay-tile protections (Fig. 54, Nos. 68 and 69).

In No. 68, where the brick was set on edge and end, the lack of stability somewhat shortened the test, a considerable amount of brick falling during the first 30 minutes.

In No. 69, with the brick laid flat, little cracking or spalling developed before failure. Fusion of the brick began between the fourth and fifth test hours, and after test the brick was found fluxed away to a depth of about ½ an inch. The uniform temperature rise in the metal indicates that the fusion of the brick did not contribute greatly to the failure of the column, the same being evidently caused by normal transmission of heat through the covering (Fig. 131, p. 292).

(*h*) GYPSUM-BLOCK PROTECTIONS

(1) STRENGTH AND POROSITY OF THE GYPSUM BLOCK.—Compression tests of the solid gypsum block used for column covering gave an average strength of 468 pounds per square inch, with maximum range in values of 22 per cent below and 40 per cent above the average. The transverse strength averaged 160 pounds per square inch, the extreme range in results of individual tests being a little higher than in the compression tests (Tables 38 and 39). The porosity was quite uniform and high, averaging 62.4 per cent, as based on the total volume (Table 37, p. 363).

(2) COMPARISON OF 2 AND 4 INCH PROTECTIONS.—A comparison in points of time to failure of 2 and 4 inch gypsum protections is given in Fig. 56, where the line connects the average results

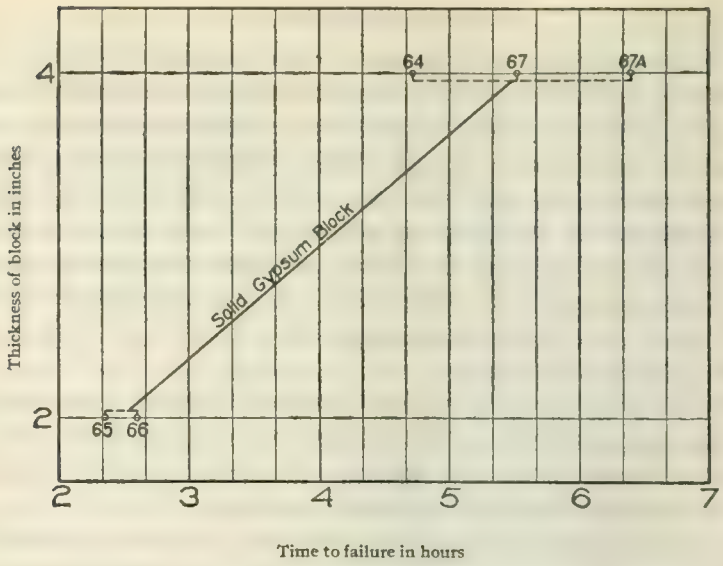


FIG. 56.—*Comparison of 2-inch and 4-inch gypsum block protections*

attained with each thickness. The 2-inch protections withstood the fire test 2 hours 22 minutes and 2 hours 36 minutes, and the 4-inch protections 4 hours 43 minutes, 5 hours 32 minutes, and 6 hours 24 minutes, respectively (Table 42*f*, p. 112.)

The protections were of solid 2 and 4 inch partition blocks set in gypsum and sand mortar, ¾ to 1¼ inch thick between blocks and column flanges, and with metal ties in the horizontal joints, the space between the blocks and the column webs being filled with gypsum block set in place or with a filling poured in place consisting of calcined gypsum, sand, and broken gypsum block.

The variations in results obtaining for each thickness of covering come within limits where they can be ascribed to incidental differences in material, workmanship, and test conditions, considering that the duration of the test was dependent upon the stability of individual blocks.

(3) CHARACTERISTIC FIRE EFFECTS.—The gypsum coverings failed due to checking, shrinking, and disintegration of the blocks which caused them to loosen and fall off. Characteristic heat effects are shown in Fig. 57. The process responsible for these effects consists mainly in the transformation of hydrated gypsum of the formula, $Ca\ SO_4+2H_2O$, to anhydrous calcium sulphate, by evaporation of the chemically combined water.

FIG. 57. *Blocks from gypsum coverings after test*

Failure of the column occurred within 20 to 40 minutes after the first blocks had fallen. The interval between maximum expansions and failure was relatively short, due to the rapid temperature rise in the steel and the low load-carrying capacity of the covering material that remained in place.

(4) HEAT-INSULATING PROPERTIES.—The maximum temperature in the steel up to the point where the blocks began to fall off was generally below 150° C (302° F), which was much lower than those obtaining in comparable tests with the other covering materials after the same duration of fire exposure (Figs. 127 to 130). The high heat-insulating value of gypsum is due in part to the heat consumed by the change in crystalline structure noted above.

(5) EFFECT OF WATER APPLICATION.—In the case of the 2-inch. protection the effect of the first 2 minutes of the water application was confined to washing away of the partly calcined gypsum near the outer face of the covering, all blocks remaining in place. During the third minute most of the blocks on the sides on which water was applied were carried down along with portions of the poured filling (Fig. 86, p. 245).

On the 4-inch protection the water application of 5-minutes duration washed away the gypsum on three sides to a depth of 1 inch from the surface, all blocks remaining in place.

(*i*) REINFORCED-CONCRETE COLUMNS

In the reinforced-concrete columns of the fire-test series the coarse concrete aggregates used were Chicago limestone and New York trap rock, and the application of the results should be limited to columns made with these concrete aggregates. Comparisons given in paragraph (*e*) above on the behavior of concrete made with these and other aggregates and applied in coverings for steel columns indicate that less favorable results would be obtained with some of them when applied in reinforced-concrete columns than was obtained with the columns tested. Also in the fire and water tests, the sections of the columns made of siliceous gravel concrete developed much greater disruptive effects during the relatively short fire exposure preceding the water application than the sections made of limestone or trap-rock concrete. The behavior of limestone and trap-rock concrete in tests of reinforced-concrete columns was similar to that of the corresponding concrete of the column coverings, little cracking or spalling of consequence occurring before failure.

(1) Influence of Concrete Aggregate.—The limestone-concrete columns all withstood the 8-hour fire test and while hot sustained loads exceeding twice the load applied in the 8-hour period. The two vertically reinforced trap-rock concrete columns failed after 7 hours 23 minutes and 7 hours 57 minutes, respectively, and the hooped column withstood the 8-hour fire test and failed under a load about 25 per cent greater than the load sustained during the fire test (p. 113). A 2-inch thickness of concrete next to the surface was assumed as covering in all cases and not included in the area used in computing working loads. The difference in results within the group can be attributed to concrete aggregate, the other incidental factors being comparable or favoring the tests giving the lower results. The trap-rock concrete fused and fluxed at some points to a depth of about 1 inch, which undoubtedly affected the time to failure to some extent. The results obtained with the concrete of both aggregates show a high degree of fire resistance.

(2) Effect of Form of Column and Reinforcement.—No effects due to shape of column or form of reinforcement were evident, differences in results being within the limits of incidental variations in test columns and test conditions.

No line of cleavage outside of the wire reinforcement was found after test in the hooped column of limestone concrete, except in the immediate region of failure, where it was apparently induced by the strains that developed when the column failed. In the case of the corresponding trap-rock concrete column, more evidence indicating separation of the outer protection from the core at the line of the reinforcement was found, effects which may in part have been caused by the fire exposure.

(3) Recovery of Strength after Fire Test.—One length of each of the hooped reinforced-concrete columns about 3 feet long was cut outside of the failure region in the fire test and subsequently tested in compression. The limestone-concrete specimen sustained a total load of 517 000 pounds as against 243 000 pounds immediately following the fire test, and the trap-rock concrete specimen 342 000 pounds, compared with 163 000 pounds at the end of the fire test. While a portion of the difference may be due to initial variations in the strength of the concrete, the greater part can be ascribed to recovery in strength of concrete and reinforcement.

(4) EFFECT OF WATER APPLICATION.—The concrete of the columns subjected to fire and water tests was placed in three sections to permit using two or three kinds in each column.

In the case of the square, vertically reinforced column, No. 111, the water carried away the concrete at the corners outside of the bars and pitted the concrete on the most exposed face to depths of from ⅛ to 1 inch for the limestone concrete and to a depth of 2 inches for the Meramec River gravel concrete in the middle section (Fig. 88, p. 247).

In the round, vertically reinforced column, the limestone concrete was pitted to a depth of 1 inch, and some of the concrete in the upper portion of the Joilet gravel concrete section was carried away. In the middle section, consisting of Meramec River gravel concrete, the outer concrete was stripped off by the water, exposing the reinforcing bars on two sides. In this, as in the preceding test, large cracks had formed in the concrete of the middle section during the fire period.

In the fire and water test of the hooped reinforced-concrete column, the water stripped the Meramec River gravel concrete and the granite concrete from the wire reinforcement on three sides during the first 15 seconds of the water period (Fig. 89). Spalling of concrete had exposed portions of the reinforcement in the middle section during the fire period. Further application of water caused stripping of reinforcement in the upper section of trap-rock concrete and increased the effects in the lower sections.

The condition of none of the reinforced concrete columns was such as to cause apprehension of early failure on being again exposed to fire after the water test, since the proportion of the load normally carried by the steel reinforcement was not large. Load tests to failure made after the water test gave factors of safety of over four as based on the calculated working load.

(*j*) TIMBER COLUMNS

Six tests of timber columns were made, four being tested unprotected. One was protected by a single layer of Portland-cement plaster on metal lath and one by a single layer of gypsum wall board. Two species of wood, longleaf pine and Douglas fir, and two types of post cap details were employed (Fig. 10).

(1) UNPROTECTED TIMBER COLUMNS.—The time to failure of the unprotected timber columns varied from 35 to 50 minutes,

failure occurring in all cases at the bearings on the steel or cast-iron cap introduced near the top of the column (Figs. 80 to 82). The deformation at the bearing increased rapidly after the first 20 minutes, and at the failure the consequent depression equaled 3 inches or more (Fig. 47, p. 126).

The average time to failure in the tests with longleaf pine was nearly the same as that obtained with Douglas fir, although the tests were hardly comparable on account of the higher moisture content of the Douglas fir in the condition tested (Table 2, p. 30).

The columns with the cast-iron cap and pintle stood up 7 minutes and 15 minutes longer than the columns with the steel-plate cap, the deformations sustained by the wood at the cap before failure ensued being larger in the case of the cast-iron bearing.

(2) PROTECTED TIMBER COLUMNS.—The outer coat of the plaster on parts of the lower half of the metal lath and plaster protection (test No. 78) spalled during the first half hour of the fire test, followed later by local buckling out of lath and supporting channels. Flames from the column issued through this opening before the end of the first hour, and about 15 minutes later flames issued from cracks in the plaster around the column cap. Crushing of the wood at the bearing due to heating of the cap caused a rapid rate of depression, beginning at 1 hour 40 minutes (Fig. 47), failure accompanied by fracture of the cap occurring at 2 hours 15 minutes (Fig. 79, p.238).

The wall board consisted of gypsum-plaster filling between paper facings. At 20 minutes flames issued from horizontal cracks in the plaster board. The board began to fall off the body of the column at about 40 minutes, and at 54 minutes nearly one-half of the covering had fallen. At 32 minutes part of the covering fell off the flanges of the steel-plate cap, the failure at 1 hour 13 minutes being due to slipping at the cap bearing same as for the unprotected timber columns (Fig. 81, test No. 80).

The protections increased the ultimate resistance of the columns by 100 to 200 per cent as compared with that in the unprotected condition. Protections on timber columns should be applied with due consideration for possible deterioration of the wood that may under certain conditions be induced by the presence of the coverings, particularly when the wood is not fully seasoned or where it is exposed to dampness.

(3) Strength After Fire Test.—The failure being localized at the bearing did not develop the full resistance of the column. The wood was burnt and charred to depths of ¼ to 1¼ inches, involving reductions in effective area of 29 to 55 per cent. One specimen 3 feet long was cut from each of the tested columns and tested in the compression about 10 weeks after the fire test. The results of the tests are given in Table 45, where also are given results of tests on comparable specimens of unburnt timber. The results indicate a reduction in the total load approximately in proportion to the reduction in area, the average of the maximum unit loads sustained by the burnt and unburnt timber of each species being nearly equal.

The strength remaining in the timber at the end of the fire test was considerably lower than the values given in Table 45, due to its heated condition.

TABLE 45.—COMPRESSIVE STRENGTH OF TIMBER AFTER FIRE TEST

Specimens 3 ft. long

Specimen No.	Species	Section Before Test		Section After Test		Reduction, Per cent	*Moisture Content, Per cent	Maximum Load	
		Outside Dimensions, In.	Area, Sq. In.	Outside Dimensions, In.	Effective Area, Sq. In.			Total, Lb.	Lb. per Sq. In.
78-1	Longleaf pine.	11.3 by 11.5	129	7.9 by 8.4	58	55	15.5	370,000	6380
79-1	Longleaf pine.	11.2 by 11.2	125	8.8 by 9.1	76	44	14.4	381,000	5010
80-1	Longleaf pine.	11.2 by 11.3	126	9.0 by 9.0	75	40	15.8	399,000	5320
81-1	Longleaf pine.	11.4 by 11.4	129	9.0 by 9.1	77	40	16.2	475,000	6170
	Longleaf pine.	Average...					15.5		5720
82-1	Douglas fir....	11.4 by 11.4	129	9.5 by 9.5	87	33	22.7	324,000	3725
83-1	Douglas fir....	11.4 by 11.4	129	10.6 by 10.7	91	29	18.8	400,000	4395
	Douglas fir....	Average...					20.7		4060
P-1	Longleaf pine.	11.2 by 11.4	127	Specimen of unburnt timber.			19.3	750,000	5705
F-1	Douglas fir....	11.4 by 11.4	129	Specimen of unburnt timber.			18.9	502,000	3890
F-2	Douglas fir....	11.4 by 11.4	129	Specimen of unburnt timber.			18.9	500,000	3875
	Douglas fir....	Average...					18.9		3880

*Determined from samples taken near point of failure, dried to constant weight at 100° C.

XIII. FIRE-RESISTANCE PERIODS DERIVED FROM THE TEST RESULTS

The results of the tests in point of time to failure will be summarized in terms of hours and minutes of fire resistance afforded by the different types of columns and protections tested.

1. BASIS OF DERIVATION

(*a*) METHOD OF COMPUTATION

A given resistance period is taken to hold, if the time to failure in the fire test or the average of the time to failure in a group of similar tests is equal to 1½ times the given resistance period. The deduction of one-third of the test duration is made to allow for incidental variations in material and workmanship of columns and coverings and differences in load and fire conditions that cause variations in results with nominally comparable columns. Individual test results within a given group may be below this limit, but not below the designated resistance period.

(*b*) INTERVALS

Resistance periods are taken 5 minutes apart up to one-half hour, at 15-minute intervals from one-half hour to 1 hour, at one-half hour intervals from 1 to 4 hours, and at 1-hour intervals from 4 to 8 hours. A variation of one-fourth of an interval is allowed, a given even value of the resistance period being taken if the period, computed according to the method given above, is not more than one-fourth of the interval below the given period value.

(*c*) TABLE OF FIRE-RESISTANCE PERIODS

A tabulation of the fire-resistance periods obtained from the results of the present series of tests is given in Table 46.

(*d*) DERIVATION OF METHOD

The method of computation is obtained from comparison of duplicate or nearly duplicate column tests considered in connection with conditions affecting comparable column constructions in buildings. In the fire tests, the maximum range in time to failure of unprotected structural steel columns was 49 per cent, comparable tests of partly protected steel columns gave a range in values of 24 per cent, plaster on metal lath protections, 15 per cent,

TABLE 46.—FIRE RESISTANCE PERIODS DERIVED FROM THE TEST RESULTS

Type of Column	PROTECTION		Minimum Area of Solid Material, Sq. In.	Nominal Thickness of Protection, In.	Fire Resistance Period
	Material	Details			
Structural steel	Unprotected	Minimum metal thickness, .20 in.	8	.....	10 min.
Structural steel, solid section	Partly protected by filling, reentrant spaces with concrete. Concrete aggregates; limestone, calcareous gravel, trap rock, granite, sandstone, or hard coal cinder	Mixture, 1:6 or 1:8. Concrete tied with vertical and horizontal steel ties	35	.....	½ hr.
Structural steel, solid section	Partly protected by filling, reentrant spaces with concrete. Aggregates; limestone, calcareous gravel, or trap rock	do.....	60	.....	¾ hr.
Structural steel, open latticed section	Partly protected by filling interior and reentrant spaces with concrete; trap rock aggregate	Mixture, 1:6. Filling extends to outside rivets and covers lattice and main members	120	.....	2 hr.
Structural steel, open latticed section	Partly protected by filling interior and reentrant spaces with concrete; limestone or calcareous gravel aggregate	do.....	120	.....	3½ hr.
Structural steel	Portland cement plaster on metal lath	Proportion of plaster, 1:$\frac{1}{10}$:2½, Portland cement, hydrated lime, and sand	40	1 layer 1 in.	¾ hr.
Structural steel	do.....	do.....	80	2 layers each ⅞ in.	1½ hr.
Structural steel	Concrete; siliceous gravel aggregate	Mixture, 1:6. Concrete tied with steel ties or wire mesh, equivalent to not less than No. 5 (B. & S. gage) wire on 8-in. pitch	100	2	1 hr.
Structural steel	do.....	do.....	200	4	2½ hr.
Structural steel	Concrete; granite, sandstone, or hard coal cinder aggregate	Mixture, 1:6. Concrete tied as above	100	2	2½ hr.
Structural steel	do.....	do.....	140	3	3½ hr.
Structural steel	do.....	do.....	200	4	5 hr.
Structural steel	Concrete; trap rock aggregate	Mixture, 1:6. Concrete tied as above	100	2	3 hr.
Structural steel	do.....	do.....	140	3	4 hr.
Structural steel	do.....	do.....	200	4	5 hr.
Structural steel	Concrete; limestone or calcareous gravel aggregate	Mixture, 1:6. Concrete tied as above	100	2	4 hr.
Structural steel	do.....	do.....	140	3	6 hr.
Structural steel	do.....	do.....	200	4	8 hr.

TABLE 46.—FIRE RESISTANCE PERIODS DERIVED FROM THE TEST RESULTS—Continued

Type of Column	PROTECTION		Minimum Area of Solid Material, Sq. In.	Nominal Thickness of Protection, In.	Fire Resistance Period
	Material	Details			
Structural steel, solid section	Hollow tile; semi-fire clay, medium hard. No filling	Mortar joint between tile and column flanges. Outside wire ties	80	2, 3, or 4	1 hr.
Structural steel, solid section	Hollow tile; surface clay or shale. Concrete filling on web sides	do	100	2, 3, or 4	1 hr.
Structural steel, solid section.	Hollow tile, extra heavy; surface clay. Concrete filling on web sides	do	160	3 or 4	1½ hr
Structural steel	Hollow tile; semi-fire clay or surface clay. Concrete filling all around	Outside wire ties	160	2, 3, or 4	2 hr.
Structural steel	do	Metal ties in horizontal joints	160	3 or 4	2½ hr.
Structural steel, solid section	Hollow tile; surface clay. Hollow tile filling	Mortar joint between tile and column flanges and webs. Metal ties in horizontal joints	240	2 layers each 2 in.	3 hr.
Structural steel, solid section	do	Mortar joint between tile and column flanges and webs. Outside wire ties	240	2 layers each 2 in.	1 hr.
Structural steel	Common brick; surface clay	Brick set on edge and end	140	2¼	1 hr.
Structural steel	do	Brick laid on side	240	3¾	5 hr.
Structural steel	Solid gypsum block. Gypsum block or poured gypsum filling	Metal ties in horizontal joints. Mortar joint between blocks and column flanges	130	2	1½ hr.
Structural steel	do	do	180	3	2½ hr.
Structural steel	do	do	240	4	3½ hr.
Round cast iron	Unprotected; unfilled	Minimum thickness of metal, .60 in.	12		20 min.
Round cast iron	Unprotected; interior filled with concrete	do	35		½ hr.
Round cast iron	Portland cement plaster on high ribbed metal lath	Proportion of plaster, 1:$\frac{1}{10}$:2½, Portland cement, hydrated lime and sand	60	1 layer, 1½ in. ½ in. air space	2 hr.
Round cast iron	Concrete; trap rock, granite, or hard coal cinder aggregate	Mixture, 1:7. Concrete tied with steel ties equivalent to not less than No. 5 (B. & S. gage) wire on 8-in. pitch	70	2	2 hr.
Round cast iron	Hollow tile; porous semi-fire clay	Outside wire ties. Mortar joint between tile and column	70	2	2 hr.

TABLE 46.—FIRE RESISTANCE PERIODS DERIVED FROM THE TEST RESULTS—Concluded

Type of Column	PROTECTION		Minimum Area of Solid Material, Sq. In.	Nominal Thickness of Protection, In.	Fire Resistance Period
	Material	Details			
Steel pipe	Unprotected. Filled with concrete	Concrete mixture, 1:1½:3	35	.....	25 min.
Reinforced steel pipe	Unprotected. Filled with concrete and reinforced in the fill with structural shapes	Concrete mixture, 1:1½:3	45	.....	¾ hr.
Reinforced concrete	Limestone or calcareous gravel concrete	Mixture 1:6. Concrete reinforced with vertical bars and lateral ties or hooping	220	2	8 hr.
Reinforced concrete	Trap rock concrete	do.....	220	2	5 hr.
Timber, longleaf pine or Douglas fir	Unprotected	Unprotected cast iron or steel cap	120	.....	25 min.
Timber, longleaf pine or Douglas fir	Gypsum wall board	Cast iron or steel cap	140	1 layer, ⅜ in.	¾ hr.
Timber, longleaf pine or Douglas fir	Portland cement plaster on metal lath	Cast iron or steel cap. Proportion of plaster, 1:½:2½, Portland cement, hydrated lime, and sand	160	1 layer, 1 in. with ¾ in. air space	1½ hr.

and 2-inch concrete protections 25 per cent, all percentages being based on the highest value within the group. The time to failure in three tests of unprotected cast-iron columns differed by less than 1 per cent. In hollow clay-tile protections, excluding Nos. 50 and 50A on account of not being tied, the maximum range in test duration within a group of comparable tests was 30 per cent, and in gypsum-block protections 26 per cent. Unprotected timber columns with exposed steel or cast-iron caps gave a range in time to failure of 30 per cent, part of which was due to difference in bearing details.

In the case of unprotected structural steel columns, the range in shape of section tested was large and a deduction of one-third from the average time to failure gives a period below which test results, obtained under comparable load and fire conditions, are not likely to fall, with the limitations in thickness of metal and area of section given in Table 46. All of the other variations noted above come within 30 per cent of the highest value in the respec-

tive groups, although it is appreciated that the test duplications were not sufficient in number to develop the full possibility of difference in results; also, that columns and protections in buildings are subject to greater variability than the test columns. The basis of derivation adopted assumes possibility of variation of about 33 per cent above and below an average value, or a total of about 66 per cent of the average. Irrespective of whether or not this range in results would fully develop in a sufficiently large number of comparable cases in buildings, the use of the one-third reduction to obtain minimum ultimate fire-resistance periods is considered justified because of the few number of tests on which most of the resistance periods are based, several of them being derived from single tests, the results of which it is necessary to assume fall within the higher rather than the average or lower range of possible variation.

(*e*) RESISTANCE TO WATER APPLICATION

With reference to fire and water exposure the results are regarded as satisfactory if the applied load was safely sustained during the fire and water periods, and in the case of the protected columns, if the covering remained in place, after a 2-minute period of water application, to such extent as to prevent early failure of the column on being again exposed to fire, precluding in all cases complete removal of the covering from any section of the column.

(*f*) SIZE LIMITATIONS

The derived fire-resistance periods apply most nearly to columns of approximately the same size as those tested and should be applied with considerable caution in connection with smaller columns. Minimum areas with relation to the derived periods are given in Table 46. The studies made on effect of size indicate that the test results can be applied with safety to columns of larger size than those tested.

(*g*) APPLICATION TO BUILDING CONDITIONS

It is believed that the types of columns and protections tested can be applied with confidence in building construction as being able to resist fires of duration equal to the pertaining resistance periods herewith derived for the respective types, provided that reasonable care is taken to identify the material used and to secure a fair grade of workmanship.

In constructing the coverings and columns, a consistent effort was made to introduce methods and conditions similar to those incident with building construction, and it is thought that the columns as tested are fairly representative of the average attained in good building practice.

The columns tested covered a wide range in material and shape of section, and each class of covering material was represented by its main varieties in common use. In applying the results broadly some difficulty may be experienced due to difference in materials of the same kind or name as occurring in different localities, relative to which some explanatory notes and cautions are given in deriving fire-resistance periods with the respective materials. It is believed that if the materials are properly identified and classed with the corresponding materials employed in the column tests, the resulting difference in fire-resisting properties will not be large, provided the mineral constituents and impurities are within the limits hereafter given.

In the column tests the temperature indicated by the pyrometers, while corresponding on the average quite closely with those on the reference curve (compare reference curve and average curve, Fig. 39), was lower than that of the furnace gases surrounding them, due mainly to radiant heat interchange between the pyrometer and the colder furnace inclosure. The difference was determined to be as large as 150° C for points near the beginning of the test, with gradual decrease to less than 50° C at the end of an 8-hour run (Sec. IX, par. 2*d*, Fig. 41). The high intensity of the furnace exposure to which the columns were subjected can be taken as compensating to some extent for differences in material incident with broad application of the test results, and give assurance that such application is justified where the variance from the materials tested is not too large.

The loads applied to the columns during test come within the higher range of normal working loads, and it is believed that the loads imposed on columns under fire conditions in buildings will seldom be much larger, either as caused by floor loads or by unequal expansion of adjacent columns during the fire.

The column coverings had a full and firm bearing at the base of the column and were full and continuous at the top, and therefore took portions of the applied load proportionate to their

relative area and rigidity with respect to the structural section. Coverings as applied in buildings will sometimes have less firm bearing at the base and be less continuous at the top, due to obstructions to proper placement offered by the floor members. This difference in ability to carry load will affect chiefly the concrete protections and fillings, since the other types have little load-carrying capacity near failure. With concrete protections it will affect only the period between maximum expansion and failure, the metal section assuming almost all of the load during the preceding period, due to its higher rate of expansion. Concrete will develop bond within a relatively short distance with a number of forms of metal section, making the covering in effect a part of the column in the middle portion of its length, even if it is not fully continuous at the ends. It is therefore apparent that the difference between the test condition and the condition of possible occurrence in buildings is not large as far as it concerns the fire resistance of the column, and considering the relatively smaller variations due to other causes generally incident with concrete protections, it can be taken as sufficiently allowed for in the reduction of the time to failure adopted for deriving fire-resistance periods.

The fire resistance afforded by columns is based on the time to failure rather than the useful limit because the former is definitely determined by the test procedure. It is deemed, however, that with the interpretation of test results given above the protection given by the columns and coverings will generally be sufficient to prevent permanent damage of such extent as to require repair or replacement after fire exposures corresponding in duration to the resistance periods here established. Only where very severe local fire exposure is coincident with other unfavorable conditions, such as inferior material and workmanship, will the resistance of the column beyond its useful limit be likely to be developed.

2. DERIVATION OF FIRE-RESISTANCE PERIODS

In deriving the resistance periods the method described in paragraph 1*a* of this section is followed, giving consideration also to effects of water application as explained in paragraph 1*e*. Interpolations for covering thicknesses intermediate between those tested are made where justified by the test results. Limited extension of results to materials not included in the particular

combination considered, but giving similar test characteristics in other related tests of this series, is made in the case of concrete and hollow clay-tile protections (Table 46, p. 197–199).

For columns with square or rectangular protections, all portions of the main members are assumed covered by not less than the nominal thickness of material specified. With round coverings, on columns of square or rectangular outline, the distance from the surface to the edge of the main column members may be reduced somewhat, provided the resulting cross-sectional area is not less than that of a square or rectangular protection of the same nominal thickness.

Details having relatively large areas, such as lattice bars and splice plates, need to be covered by the nominal thickness specified. For smaller areas as of rivets and extreme edges of bracket or supporting angles, the covering thickness may be reduced to 1 inch.

Where partitions or pipe inclosures are supported by column coverings the connection should be made so that the stability of the covering is not affected by buckling or failure of the supported walls. Pipes should not be placed within the thickness of covering required for the given resistance periods nor incorporated in the covering in such manner as to dislodge any essential portion thereof by expansion or buckling incident with normal use or with fire conditions.

(*a*) UNPROTECTED STRUCTURAL-STEEL COLUMNS

As based on test Nos. 1 to 8, and with not less than the minimum metal thickness and area of section given in Table 46, unprotected structural-steel columns give *10-minute* fire resistance.

The low degree of resistance afforded indicates unsuitability for use where fires of any appreciable degree of intensity and duration are possible.

(*b*) PARTLY PROTECTED STRUCTURAL-STEEL COLUMNS

The protection consists of concrete that fills the interior and reentrant portions to the outside of the extreme metal on the main members. Except on the open sides of latticed columns, the concrete is secured with horizontal and vertical metal ties.

(1) SOLID-SECTION COLUMNS.—Structural-steel columns of solid rolled or riveted section, whose reentrant portions are filled with

concrete made with any of the aggregates included in the present series of tests, except highly siliceous sand and gravel, mixed in proportion of 1 part Portland cement to not more than 8 parts fine and coarse aggregates combined, and with combined area of steel and concrete not less than 35 square inches, give *one-half hour* fire resistance. Similar columns filled in the same manner with limestone, calcareous gravel, or trap-rock concrete, and with area of filled column not less than 60 square inches, give *three-fourths hour* fire resistance.

The one-half hour period is based on test Nos. 15, 16, and 17, and the three-fourths hour period on test Nos. 14, 20, and 21.

(2) OPEN LATTICED SECTION.—Structural-steel columns of open latticed section, with the interior and reentrant portions filled with concrete of proportion 1 part Portland cement to not more than 6 parts fine and coarse aggregates combined, and with total area of not less than 120 square inches, give *2-hour* fire resistance if concrete is made with trap-rock aggregate and *3½-hour* fire resistance if limestone aggregate is used.

The above conclusions are based on test Nos. 18 and 22, the higher resistance over the solid section columns being due to absence of large areas of exposed metal. It is essential to use concrete that is not subject to cracking or spalling, since the covering on parts of the main members is usually little more than 1 inch thick and not tied.

(c) STRUCTURAL-STEEL COLUMNS WITH PLASTER ON METAL-LATH PROTECTIONS

The plaster can be applied on either expanded metal or woven-wire lath. The layer thickness is measured from the inner line of the lath. The proportion of the plaster considered is 1 part Portland cement to not more than 2½ parts medium-coarse sand, with lime added equal to one-tenth of the cement volume. The periods derived are based on test Nos. 23 to 26 and 110, in which the above proportion was used. The effect on fire-resistive properties produced by varying the lime content is otherwise not known.

(1) SINGLE-LAYER PROTECTION.—Structural-steel columns protected by a layer of Portland-cement plaster 1 inch thick applied on metal lath give *three-fourths hour* fire resistance.

(2) DOUBLE-LAYER PROTECTION.—Structural-steel columns protected by two layers of Portland-cement plaster each ⅞ inch thick with ¾-inch air space between layers give *1½-hour* fire resistance.

(*d*) CONCRETE PROTECTIONS ON STRUCTURAL-STEEL COLUMNS

The protections, preferably placed without construction joints, should extend through floors, where the floor filling is of other material than concrete. Lateral ties consisting of iron or steel wires or mesh, embedded in the covering and extending to within about 1 inch from its surface, are to be equivalent to not less than a No. 5 (B. and S. gage) steel wire wound spirally on a pitch of 8 inches.

The proportion of mixture taken to hold is 1 volume part Portland cement to 6 parts fine and coarse aggregates combined, 94 pounds of the cement being taken as its weight per cubic foot. The effect of reducing the cement content of the mixture was not definitely determined in the tests, although some reduction in fire resistance was noted as between the 1 : 6 and the 1 : 8 proportion, estimated as equivalent to reductions in the resistance periods of not more than ½ hour for periods up to 3 hours and possibly as much as 1 hour for the longer resistance periods.

Limitations relative to size are given in Table 46 (p. 197). For columns of dimensions much larger than the given minimum, the periods apply with a considerable margin of safety.

(1) SILICEOUS-GRAVEL CONCRETE PROTECTION.—Structural-steel columns covered with a 2-inch thickness of concrete made with gravel aggregate of high silica content give *one-hour* fire resistance, and with similar 4-inch protections give *2½-hour* fire resistance.

Since the failure of coverings of this type of concrete is dependent on stability of portions thereof, no adequate basis is present for estimating the value of the resistance period for thicknesses intermediate between 2 and 4 inches.

The periods are based on test Nos. 39 and 45, where the aggregate consisted mainly of chert grains and pebbles. Since this form of silica develops as great or greater disruptive effects on exposure to fire as quartz, flint, and related minerals, the derived protection periods apply generally where concrete made with all common forms of siliceous aggregates is used.

(2) GRANITE, SANDSTONE, OR HARD-COAL CINDER CONCRETE PROTECTION.—Structural-steel columns covered with 2-inch protections of granite, sandstone, or hard-coal cinder concrete give *2½-hour* fire resistance, with 3-inch coverings give *3½-hour* fire resistance, and with 4-inch coverings give *5-hour* fire resistance.

Tests were made of 2-inch coverings of concrete made with sandstone and cinder (Nos. 31, 32, 32A, 43, 44, and 104), and in the 4-inch thickness, with granite as the coarse aggregate (test Nos. 34, 34A, and 103). The extension of the results with sandstone and cinder concrete to cover thicknesses heavier than 2 inches, and of granite concrete to cover thicknesses lower than 4 inches, is based on relative effectiveness with respect to trap-rock concrete in coverings (test Nos. 29, 36, and 37) of comparable thickness.

The granite used in the concrete of the column coverings tested contained 35 per cent quartz and about 60 per cent feldspars, chiefly orthoclase (Table 20, p. 345), the size of crystals ranging from 0.03 inch to 0.35 inch. The range in quartz content of granites is generally from 20 to 40 per cent, with extremes from 5 to 50 per cent. In crystal size the general range is from 0.02 inch to 0.40 inch, with extremes up to 2½ inches. It appears, therefore, that the granite tested came within the higher range in quartz content and had crystals of average size. To what extent changes in mineral composition and crystal size of granite affect the fire-resistive properties of concrete made with it is not known, although it would be expected that those of lower quartz content will be the least subject to destructive fire effects.

The Berea sandstone used in these tests was a silica cemented stone of average hardness consisting almost wholly of subangular grains of quartz. Some disintegration from heat due to loss of cementing properties was noted, but not enough to affect the fire-resisting properties, the principal fire effects being cracking and dislodgment of relatively large masses, induced presumably by the high quartz content.

Among sandstones used as concrete aggregate, quartzite has all pore spaces filled with silica cement, resulting in a hard, homogeneous, quartz mass. Flame tests made on small specimens of the stone do not indicate any greater disruptive effects than those incident with similar tests of other sandstones.

Silica cemented sandstones range in hardness from compacted sand to a condition approaching quartzite, all being of high silica content, usually over 95 per cent. The softer grades are generally eliminated from use as concrete aggregate on the score of deficient strength and hardness.

The lime-cemented sandstones range in mineral composition from nearly pure sandstone to sandy limestones carrying only small amounts of silica. The iron (limonite) and clay cemented sandstones, of which the brownstones are representative, carry variable amounts of feldspar and mica, the range in free quartz content being from 50 to 90 per cent. The bluestones of New York, Pennsylvania, and West Virginia contain from 20 to 60 per cent quartz, the other minerals being mainly feldspar and hornblende.

While not possible to predict fully the behavior of combinations of minerals on exposure to fire, it is probable that the addition of calcareous, clayey, or feldspathic minerals to quartz to form the sandstones noted above will decrease rather than increase the cracking incident with fire exposure of purer forms of the rock.

The hard-coal cinder contained about 10 per cent unburned coal and 5 per cent ash, and to obtain comparable results these percentages should not be greatly exceeded.

(3) Trap-Rock Concrete Protection.—Structural-steel columns protected by 2-inch coverings of trap-rock concrete give *3-hour* fire resistance, with 3-inch coverings give *4-hour* fire resistance, and with 4-inch coverings give *5-hour* fire resistance (test Nos. 29, 36, 37, 40, 101, 103, and 104).

The periods for the 2 and 4 inch covering thicknesses were derived directly from the test results, and that for the 3-inch thickness taken as their average.

Trap rock is a dark-colored, fine-grained igneous rock that does not vary greatly in mineral composition and carries at most only a trace of quartz (Table 20). The term does not include the hard sandstones known under the same name in some localities.

(4) Limestone or Calcareous-Gravel Concrete Protection.—Structural-steel columns protected by 2-inch coverings of limestone or calcareous-gravel concrete give *4-hour* fire resistance, with 3-inch coverings give *6-hour* fire resistance, and with 4-inch coverings give *8-hour* fire resistance.

Tests were made of 2-inch coverings of limestone and calcareous-gravel concrete (test Nos. 28, 28A, 30, 38, 101, 102, and 104) and of 4-inch coverings of 1:6 and 1:8 limestone concrete (test Nos. 33, 33A, 35, 41, 42, and 103). The period for the 2-inch thickness was derived directly from the test results. The columns with the

4-inch covering all withstood the 8-hour fire test, and while hot sustained such large additional loads as to justify the conclusion that in the lower range of results with comparably protected columns the working load will be sustained during an 8-hour fire period.

The limestone and the calcareous gravel of the tests were dolomitic and contained about 5 per cent of clayey impurities (Table 20). It is thought that the results apply without modification where high calcium limestone or gravel of nearly the same purity is used. The silica impurities in limestone occur as free silica (quartz, chert, flint, and opal) and silica combined in clay and other silicates. Most limestones carry from 5 to 10 per cent of combined clayey impurities, but not many have more than 10 per cent of free silica, and within the given limits these impurities are not deemed objectionable for stone or clean gravel used as concrete aggregate in fire-resistive construction. Some limestones contain higher percentages of free silica and particularly the waste from limekiln and cement plant quarries is likely to be high in chert.

Gravels in the glaciated area, which includes nearly all of the territory north of the Missouri and Ohio Rivers, New England, New York, northern Pennsylvania, and northern New Jersey, may contain material from any point to the northward and formed under the most diverse conditions will vary greatly in mineral composition both vertically and horizontally. This also applies to the gravels of the Great Lakes and the ocean gravels as far south as New York. If the superior fire-resistive qualities of concrete made with calcareous gravel is to be assured, or the highly siliceous gravel avoided, it is necessary to identify the material in each bank by means of suitable chemical or mineralogical analyses at successive times as the development proceeds.

Outside of the glaciated area, the river and shore gravels are more uniform, although varying with changes in the rock formations of the drainage area or headlands.

The calcareous content of the sands (Fox River and Joliet) used in combination with the limestone and calcareous gravel was higher than the combined quartz and chert content (Table 20). While all sands are likely to contain considerable silica, and the extent of their influence on the thermal properties of concrete made with them is not known, it is thought best to avoid using highly siliceous

sands in combination with limestone and calcareous gravel where the full fire-resistive value of the concrete is to be utilized. As used with the other coarse concrete aggregates, the mineral composition of the sand is of minor importance, provided it meets the common requirements for a good concrete sand. The sand in a given deposit is generally more uniform in composition than the larger-sized material.

(*c*) HOLLOW CLAY-TILE PROTECTIONS ON STRUCTURAL-STEEL COLUMNS

The tile is assumed applied in courses about 12 inches high with fairly full horizontal and vertical mortar joints between the tile units. The mortar should also fill the space between the tile and column flanges except where it is filled with concrete. The proportion of mortar used in the tests was 1 volume part Portland cement to 1 of stiff slaked lime, and 4 of fine sand. While the influence of mortar proportion on the effectiveness of the covering is not definitely known, the test results are not fully applicable unless mortar having about the same content of cementing materials as that given above is used, and particularly the Portland cement should not be reduced below the proportion given. As an aid in assigning fire-resistance periods for combinations of tile and details of application not included in the tests, it can be stated that in the tests the semifire-clay tile burnt to medium hardness displayed the best fire-resistance properties, followed in order by tile of hard burnt semifire clay, of surface clay, and of shale.

The coverings tested were generally tied with one No. 12 (B. & S. gage) iron or steel wire placed tightly around the outside of each course, or by strips of woven wire of ⅜-inch mesh, placed in the horizontal joints and lapping at the corners. The latter method is considered superior to the outside wire ties and can be substituted for them where desired. The difference in effectiveness of the two methods is not considered so large as to justify requiring the mesh except for tile subject to severe disruptive effects on exposure to fire and where the higher resistance periods are to be attained. Other forms of interior ties may be used if equivalent in effect to the wire mesh.

In assigning fire-resistance periods to given protections, consideration is given to the effect of area of solid material in the cross section of the covered column, having reference to the minimum values given in Table 46 (p. 198).

The proportion of mixture for the concrete filling was 1 part Portland cement to not more than 8 parts fine and coarse aggregates combined, the filling being placed after the tile was set. Any aggregate used in these tests may be employed except highly siliceous gravel.

(1) UNFILLED PROTECTION.—Structural-steel columns with hollow clay-tile protections of 2, 3, or 4 inch semifire-clay tile of medium hardness, tied with outside wire ties, give *one-hour* fire resistance.

This period is based on results of fire tests Nos. 48 and 49 taken together with fire and water tests Nos. 105 and 107. The tile in these tests developed fewer disruptive effects on exposure to fire than any other kind tested. Tile more sensitive to abrupt temperature change is not adapted for use in unfilled protections, as the cracking and spalling resulting after a short fire exposure allows the heat to readily reach the steel, also, on application of water, parts of the impaired covering are liable to be carried away, leaving the metal unprotected against a possible recurring fire.

The use of unfilled protections should generally be avoided as the filling materially increases their stability and insulating value.

(2) SHALE OR SURFACE-CLAY TILE PROTECTION WITH CONCRETE FILLING REENTRANT SPACES.—Structural-steel columns with hollow clay-tile protections of 2, 3, or 4 inch shale or surface-clay tile, tied with outside wires and with concrete filling the reentrant spaces between tile and column web, give *one-hour* fire resistance (test Nos. 50, 50A, 52, 53, and 106).

Similar protections of 3 or 4 inch surface-clay tile with heavy shells and webs give *1½-hour* fire resistance (test Nos. 51 and 51A).

(3) SEMIFIRE-CLAY OR SURFACE-CLAY TILE PROTECTION WITH FULL CONCRETE FILLING.—Structural-steel columns with hollow-tile protections of the given types of clay, set 1 inch away from the flanges and edges, with concrete filling all spaces between the steel and the tile, give *2-hour* fire resistance where outside wire ties are used (test Nos. 54, 55, and 57), and with wire mesh in the horizontal joints give *2½-hour* fire resistance (test Nos. 56 and 77).

The thickness of tile, using outside wire ties, may be either 2, 3, or 4 inches. With the wire mesh the thickness should preferably be not less than 3 inches to allow ties of sufficient width to be used.

(4) DOUBLE 2-INCH TILE PROTECTION.—Structural-steel columns with protections of two layers of 2-inch surface-clay tile, and with the space between the inner tile layer and the column filled with tile set in place, give *3-hour* fire resistance when the covering is bonded with wire mesh in the horizontal joints and *1-hour* fire resistance if tied with outside wire ties.

The periods are based on test Nos. 58 and 59, which indicated a marked difference in results due to the method of tying the surface-clay tile applied in the given type of protection. Concrete filling may be substituted for the tile filling if desired.

(*f*) BRICK PROTECTIONS

The fire resistance periods are derived from the results of test Nos. 68 and 69. The same qualifications relative to mortar apply as for the hollow clay-tile protections, as also the size limitations given in Table 46 (p. 198).

Structural-steel columns thus protected with common surface-clay brick, set on edge and end outside of the flanges and edges and filling the whole space to the steel, give *1-hour* fire resistance, and with brick laid flat on side to form a solid protection of about 4-inch thickness *5-hour* fire resistance is developed.

(*g*) GYPSUM-BLOCK PROTECTIONS

The derived resistance periods apply to columns covered with solid gypsum block set in mortar of proportion, 1 part calcined gypsum to 3 parts fine sand, the blocks being bonded with strips of corrugated iron or wire mesh placed in the horizontal joints over all vertical joints. The space between the outer blocks and column can be filled with gypsum blocks set in place or with poured filling consisting of calcined gypsum, broken gypsum blocks, and sand, mixed with enough water to enable proper placement.

Structural-steel columns with protections of 2-inch solid gypsum blocks placed according to the above details give *1½-hour* fire resistance (test Nos. 65, 66, and 108), and with similar protections of 4-inch solid gypsum blocks give *3½-hour* fire resistance (test Nos. 64, 67, 67A, and 109). Interpolation between the two thicknesses appears justified, the protection made with the 3-inch solid block giving accordingly a resistance period of *2½ hours*.

The area limitations given in Table 46 apply less rigidly to gypsum-block protection than to most other types, because the

failure is induced by loss of stability of the blocks after a given fire exposure and not to normal transmission of heat through the covering.

(*h*) CAST-IRON COLUMNS

(1) UNPROTECTED COLUMNS.—With the minimum area and wall thickness given in Table 46, unprotected cast-iron columns give *20-minute* fire resistance.

The fire tests (Nos. 9. 10, and 10A) were made on round columns of about ¾-inch wall thickness, and gave uniform results, although they were not sufficient in number to develop possible variations due to quality of metal and differences in wall thickness. The fire and water tests, while inducing large permanent deflections, did not cause the columns to lose ability to sustain working loads.

Filling the interior with concrete (test No. 11) gives an increase in resistance of 10 minutes or a total period of *one-half hour.*

(2) PLASTER ON METAL-LATH PROTECTION.—A single heavy layer of Portland-cement plaster of 1½-inch average thickness applied on high-ribbed metal lath to round cast-iron columns gives a fire resistance of *2 hours* (test No. 27).

The same proportion of plaster applies as given above under (*c*) for protections on steel columns. To attain the required average layer thickness, the surface of the covering will extend about 2 inches outside of the metal on account of voids inside of the lath.

(3) CONCRETE PROTECTION.—Round, cast-iron columns with 2-inch protections of trap-rock, granite, or hard-coal cinder concrete, give *2-hour* fire resistance.

The period is based on test No. 47 where cinder concrete was used. Extension to include concrete made with the other aggregates is made because of characteristics similar to cinder concrete developed by them in other tests of concrete protection. Limestone and calcareous gravel concrete would on the same basis give longer resistance periods. The proportion of the concrete can be taken as 1 part Portland cement to not over 7 parts aggregate. Relatively small-sized aggregates need to be used, to enable good placement of the covering, and with some aggregates it may be necessary to increase the thickness of the covering to secure this result. The ties, equivalent to not less than No. 5 (B. & S. gage) steel wire on 8-inch vertical pitch, are to be supported about 1 inch away from the column.

(4) HOLLOW CLAY-TILE PROTECTION.—Cast-iron columns covered with 2-inch curved porous semifire-clay tile, with ¾-inch mortar thickness between tile and column, and tied with outside wire ties, give *2-hour* fire resistance (test Nos. 62 and 63).

The same details of application can be taken to hold as for hollow-tile protections on structural-steel columns.

(*i*) UNPROTECTED PIPE COLUMNS

Unprotected columns consisting of steel or wrought-iron pipes not smaller than the standard 6-inch pipe size, with filling of concrete, give *25-minute* fire resistance.

Pipe columns not smaller than the 7-inch standard pipe size, filled with concrete and reinforced in the fill with structural shapes, give *three-fourths hour* fire resistance.

The proportion of the concrete filling is taken to be 1 part Portland cement to not more than 4½ parts fine and coarse aggregates combined (Table 46, p. 199).

The conclusions relative to pipe columns are based on test Nos. 12 and 13, made, respectively, on a 7-inch plain pipe column and on an 8-inch reinforced pipe column, both being standard new steel pipes with wall thickness of about 5/16 of an inch. Extension of the result with the former to make it applicable to the 6-inch pipe size and of the latter to apply to the 7-inch size, appears justified.

(*j*) REINFORCED-CONCRETE COLUMNS

The proportion of the concrete for which the derived periods hold is 1 part Portland cement to 6 parts fine and coarse aggregates combined, the coarse aggregates being trap rock or limestone (cf. Sec. XII, par. 3*i*). The same considerations relative to possible variations in the aggregate hold as for concrete protection.

The columns may be round or square and reinforced with vertical bars held by lateral ties spaced not farther apart than 12 inches or by vertical bars and spirally wound wire hooping, all reinforcements to be covered by not less than 2 inches of concrete placed integrally with the structural portion of the column.

Under these conditions reinforced-concrete columns made with limestone or highly calcareous gravel-aggregate can be taken as giving *8-hour* fire resistance (test Nos. 70, 72, 74, 111, and 112), and made with trap-rock aggregate, *5-hour* fire resistance (test Nos. 71, 73, 75, and 113) (Table 46, p. 199).

The limestone-concrete columns withstood the 8-hour fire test, and while hot sustained ultimate loads of from a little less than twice to nearly three times the load sustained during the 8-hour period. This gives reasonable assurance that all similarly constructed columns, using a properly identified limestone aggregate, will sustain working load until the end of an 8-hour fire period, taking into account all probable variations. Calcareous gravel can be used in place of limestone if the free silica and other impurities are within the limits given in paragraph (*d*4) (p. 208).

Two of the trap-rock concrete columns failed during the 8-hour period and the other withstood the 8-hour fire test and about 25 per cent additional load at the end of this period. The fire-resistance period is therefore almost fully determinate on the basis of test duration.

(*k*) TIMBER COLUMNS

With unprotected or protected timber columns having exposed cast iron or steel caps, the resistance to fire is limited by failure at the bearings before the full resistance of the timber has been developed elsewhere. (Cf. Sec. XII, pp. 167 and 194). The resistance periods apply most fully to the species tested, long leaf pine and Douglas fir, although with failure occurring at the bearings the species of timber is not a governing consideration.

(1) UNPROTECTED TIMBER COLUMNS.—As based on test Nos. 79, 81, 82, and 83, unprotected timber columns with exposed cast iron or steel caps give *25-minute* fire resistance.

(2) PROTECTED TIMBER COLUMNS.—Timber columns having cast iron or steel caps and protected by a layer of gypsum wall board ⅜ inch thick, give *three-fourths hour* fire resistance (test No. 80), and protected by a 1-inch layer of Portland-cement plaster on metal lath, with an air space between the timber and the plaster layer, give *1½-hour* fire resistance (test No. 78).

The protections are assumed to cover all exposed metal of the caps as well as the timber. The proportion of the Portland-cement plaster can be taken the same as given for plaster coverings on steel columns (Table 46, p. 197).

Protections on timber columns should be applied with caution on account of possible unfavorable effects upon the timber in point of susceptibility to decay. (Cf. Sec. XII, p. 194).

3. CONDITIONS GOVERNING FIRE DURATION IN BUILDINGS

The intensity and duration of building fires depend upon the character and amount of combustible material in the building construction itself and in the building contents. The latter will be determined in a general way by the occupancy. Office and residence occupancies generally support fires of shorter duration than manufacturing, merchandizing, or storage. The possible duration of fire is also determined to some extent by the floor load for which the building is designed, since it limits the amount of material subject to fire.

The effective fire duration, as far as it concerns a given building member, pertains only to the duration of the combustion taking place near enough to it to impart temperatures sufficiently high to cause failure under its supported load. This is not necessarily the same as the total duration of the fire within the building, or within a subdivision thereof, such as a building story. It is necessary to assume maximum probable conditions both with regard to building contents and air supply, as considered with respect to intensity and duration of a possible fire. Compensations and adjustments between intensity and duration may be necessary under some conditions in order to approximate a fire duration having intensity equivalent to that of the exposure in the fire test. Limitation of the degree of resistance to be developed to the requirements of present or immediate future occupancy is generally not justified, unless the building, its location or use is such that no change to a use involving a more severe fire condition is probable.

While it is outside of the scope of the present paper to attempt the determination of probable fire duration for various types of buildings and occupancies, it is a step in the application of the test results that will require careful consideration. The fire-resistance periods were derived under exacting conditions in point of test columns, test conditions, and interpretation of results, and the types of columns and protections to which they pertain are deemed to be fully adequate for resisting building fires of corresponding duration.

XIV. APPENDIXES

APPENDIX A.—Views of Columns

Fig. No.	Test No.	Protection	Page
		BEFORE AND AFTER FIRE TEST	
58	1 to 5	Unprotected structural-steel columns	217
59	6 to 8	do	218
60	9 to 13	Unprotected cast-iron and pipe columns	219
61	14 to 18	Partly protected structural-steel columns	220
62	19 to 22	do	221
63	23 to 27	Plaster on metal-lath protections	222
64	28A to 31	Concrete protections	223
65	32 to 34A	do	224
66	35 to 39, 45	do	225
67	40 to 44, 46, 47	do	226
68	48, 49, 50	Clay-tile protections	227
69	50A, 51, 51A	do	228
70	52, 53, 54	do	229
71	55 to 58	do	230
72	59, 60	do	231
73	61, 62, 63	do	232
74	64, 65, 66	Gypsum-block protections	233
75	67, 67A	do	234
76	68, 69	Brick protections	235
76	70	Reinforced-concrete column	235
77	71 to 75	Reinforced-concrete columns	236
78	76, 77	Plastered clay-tile protections	237
79	78	Longleaf pine column, protected	238
80	79	Longleaf pine column, unprotected	239
81	80	Longleaf pine column, protected	240
81	81	Longleaf pine column, unprotected	240
82	82, 83	Douglas fir columns, unprotected	241
		BEFORE AND AFTER FIRE AND WATER TEST	
83	101, 102, 103	Concrete protections	242
84	104	Concrete protection	243
84	105	Clay-tile protection	243
85	106, 107	Clay-tile protections	244
86	108, 109	Gypsum-block protections	245
87	110	Plaster on metal-lath protection	249
88	111, 112	Reinforced-concrete columns	247
89	113	Reinforced-concrete column	248
89	114, 115	Cast-iron columns	248

FIG. 58. *Unprotected structural steel columns, test Nos. 1 to 5, after fire test*

No. 6.—I beam and channel No. 7.—Latticed angle No. 7.—Detail of failure No. 8.—Starred angle

FIG. 50.—*Unprotected structural steel columns, test Nos. 6 to 8, after fire test*

No. 9.—Cast iron, ends restrained

No. 10.—Cast iron, ends not restrained

No. 10A.—Cast-iron pipe

No. 11.—Cast iron, concrete filled

No. 12.—Steel pipe, concrete filled

No. 13.—Reinforced-steel pipe, concrete filled

FIG. 60.—*Unprotected cast iron and pipe columns, test Nos. 9 to 13, after fire test*

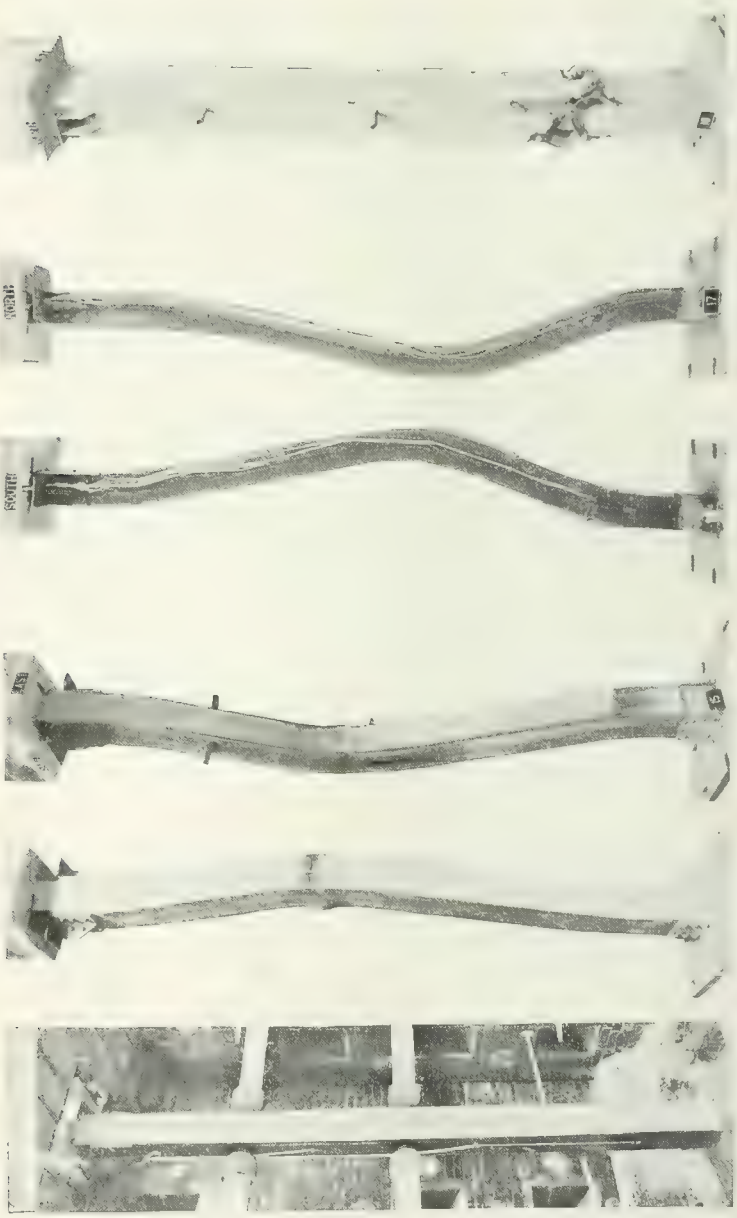


FIG. 61.—*Partly protected structural-steel columns, test Nos. 14 to 18, after the test*

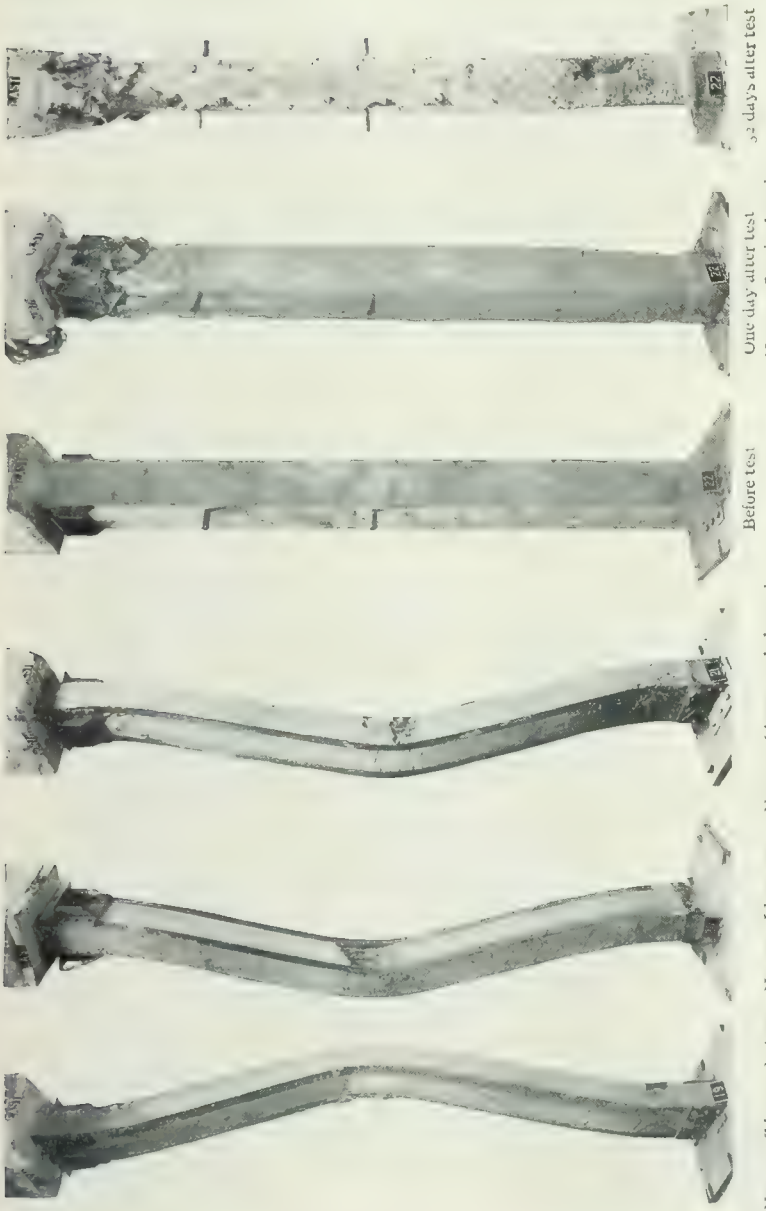


FIG. 62.—*Partly protected structural-steel columns, test Nos. 19 to 22, after fire test*

FIG. 93.—*Plaster on metal-lath protections, test Nos. 23 to 27, after fire test*

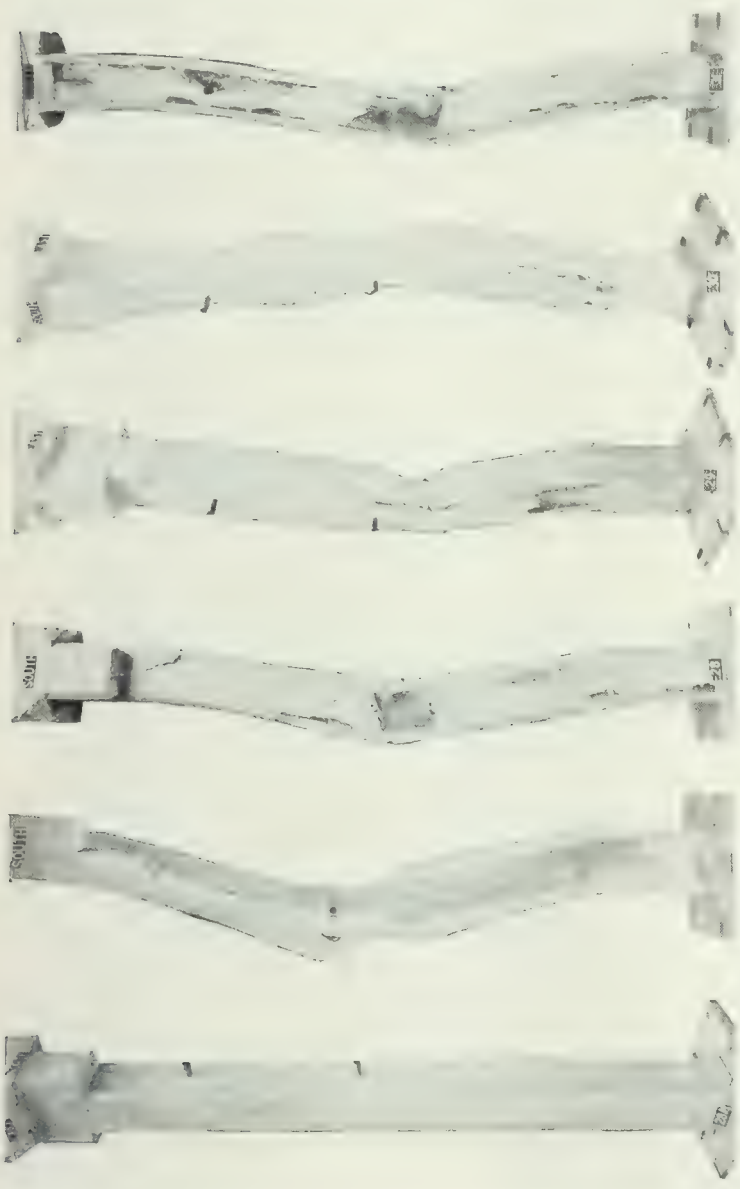


FIG. 64.—Concrete protections, tied Nos. 8A.1 to 31, after fire test

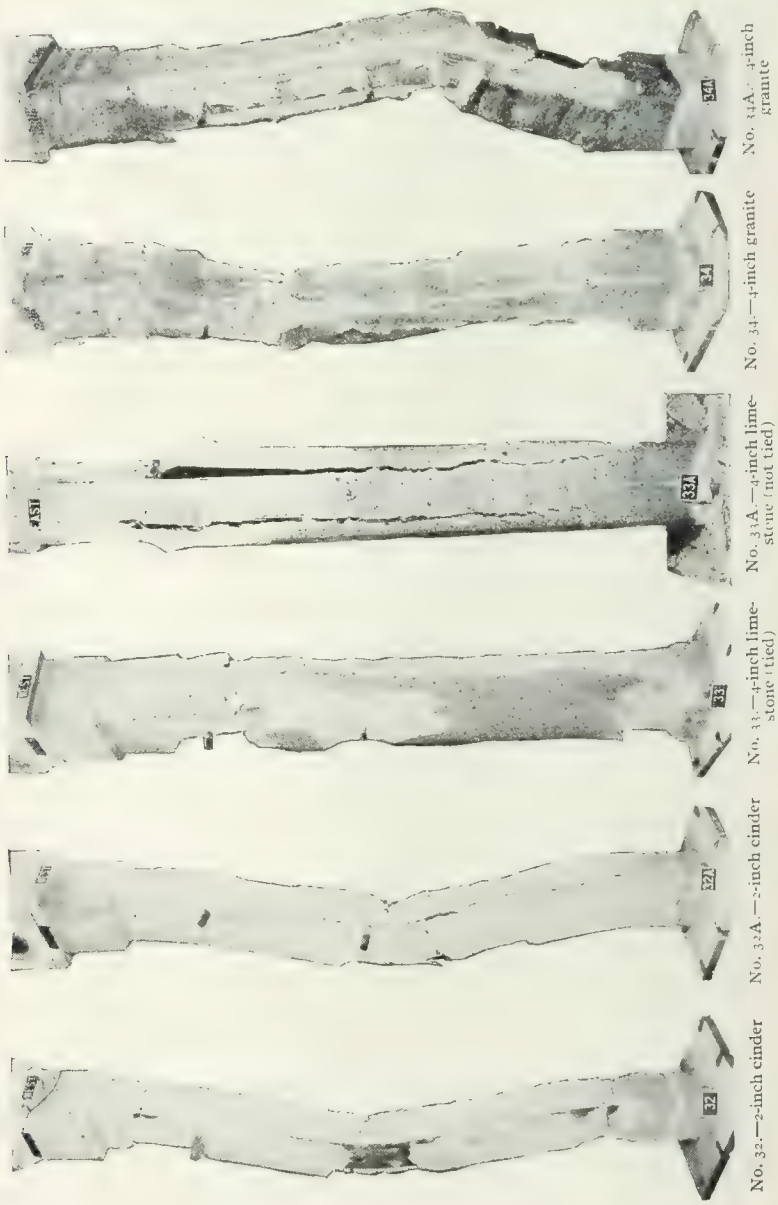


FIG. 65—*Concrete protections, test Nos. 32 to 34A, after fire test*

No. 35.—4-inch lime-stone (1 3 5)
No. 36.—4-inch trap rock
No. 37.—4-inch trap rock
No. 38.—4-inch Joliet gravel
No. 39.—4-inch Merrimac River gravel
No. 45.—4-inch Merrimac River gravel

FIG. 66.—*Concrete protections, test Nos. 35 to 39, and 45, after fire test*

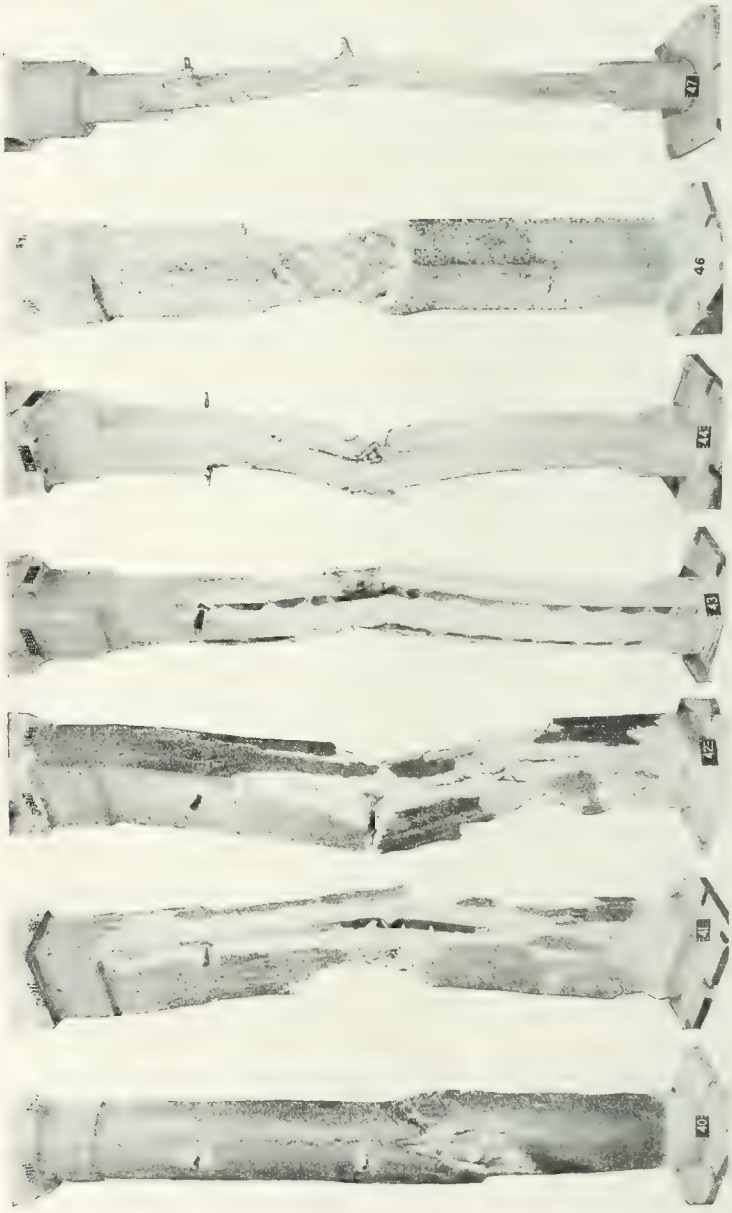
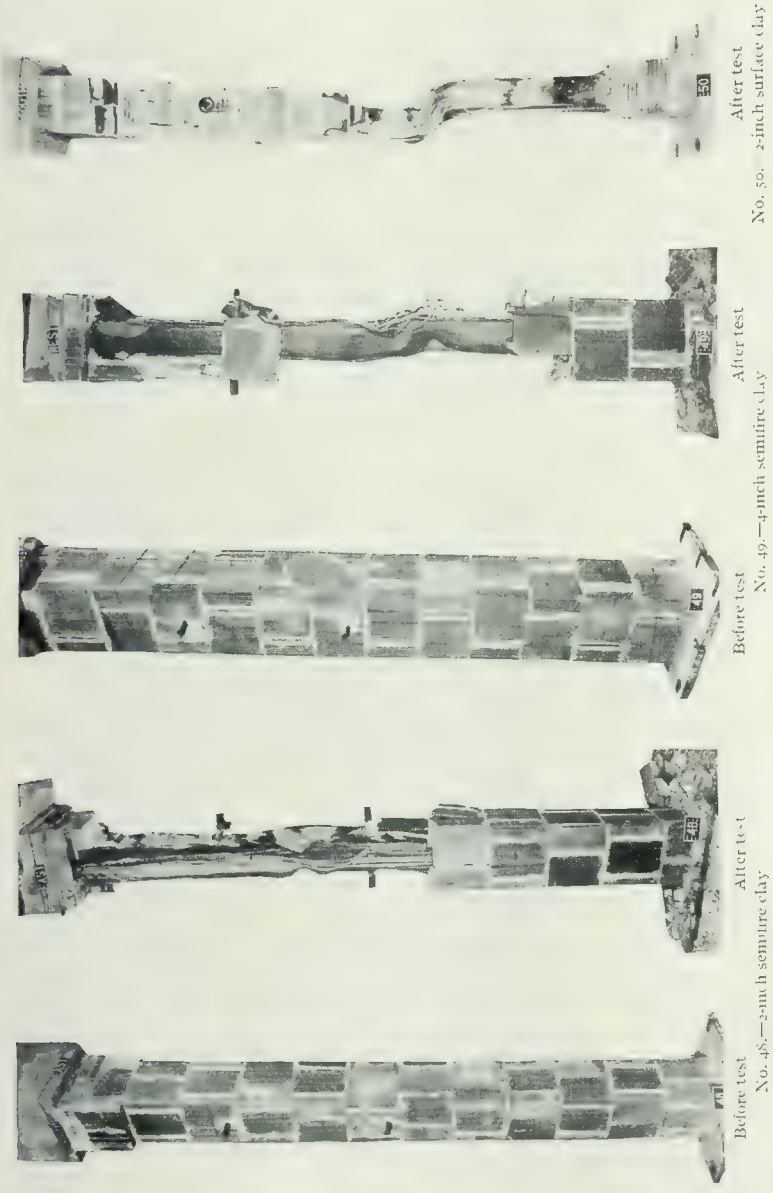


FIG. 67.—*Concrete protections, test Nos. 40 to 44, 46, and 47, after fire test*



Before test
No. 48.—2-inch semiire clay

After test

Before test
No. 49.—4-inch semifire clay

After test

After test
No. 50. 2-inch surface clay

FIG. 68.—*4 lay-tile protections, test Nos. 48, 49, and 50, before and after fire test*

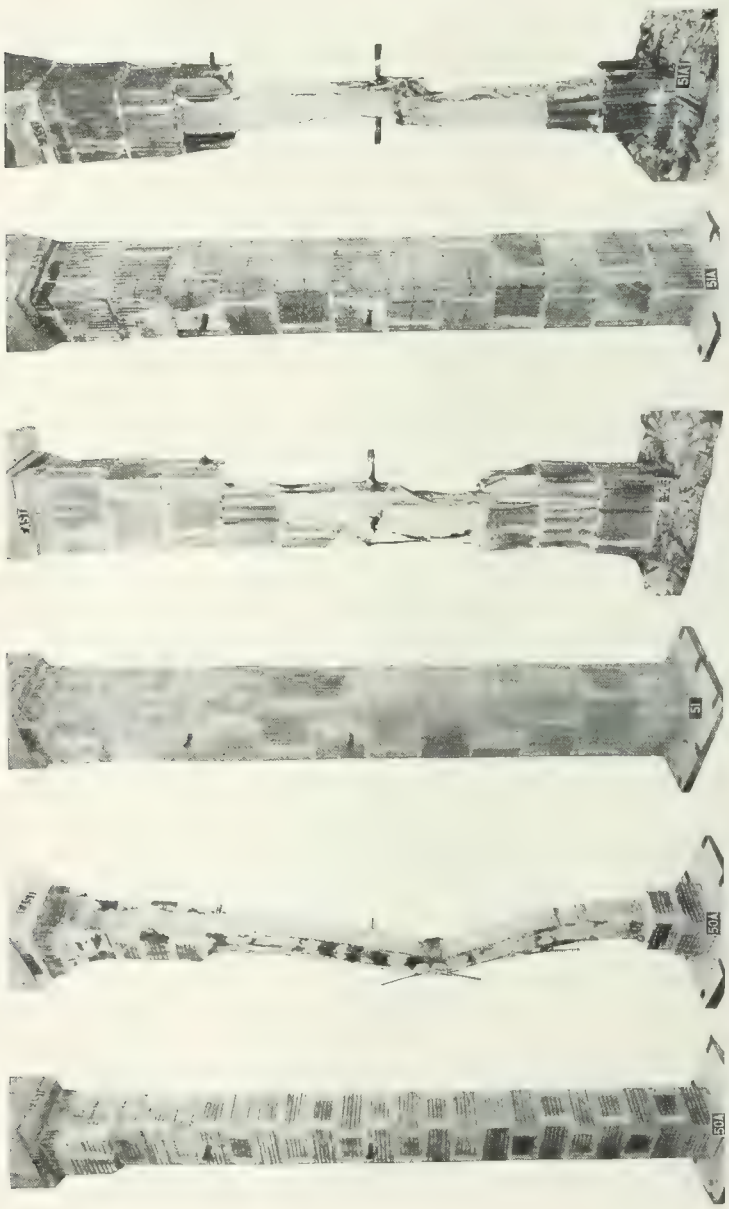


FIG. 69.—*Clay-tile protections, test Nos. 50A, 51, and 51A, before and after fire test*

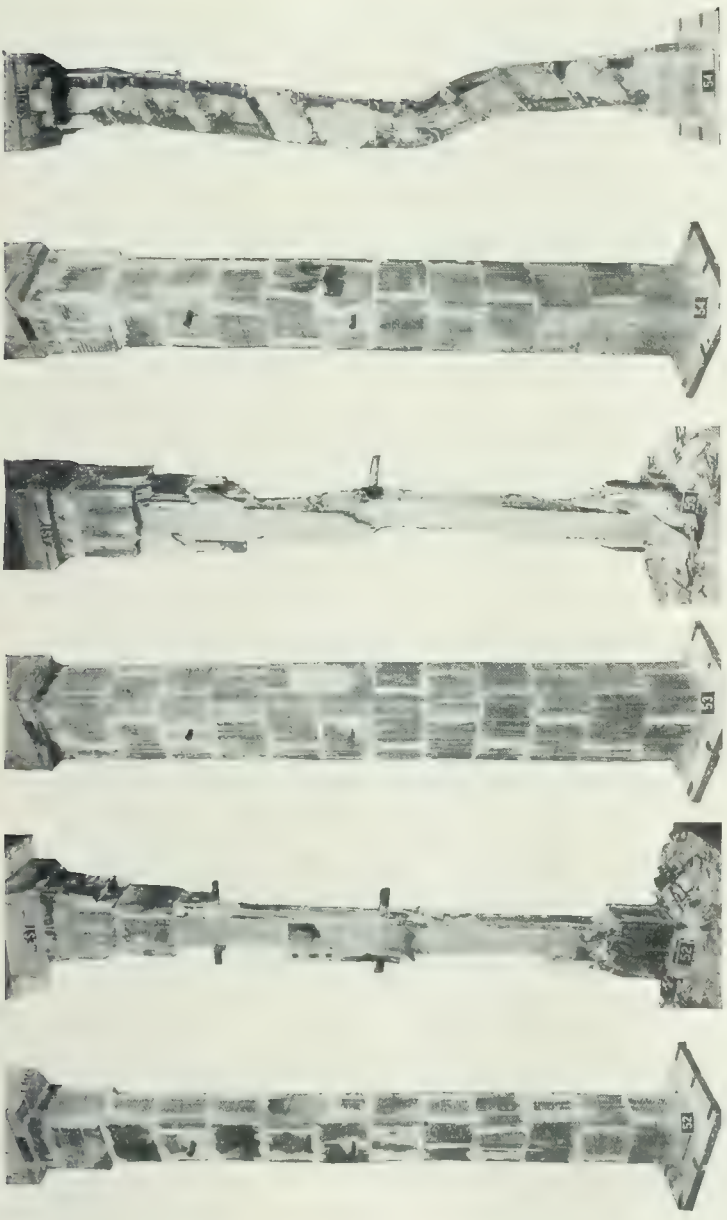

After test
Before test
No. 54—Gault brick, 4 inches thick, 4 inches solid, 4 inches cement clay
After test
Before test
No. 53—Gault brick, 4 inches thick, 4 inches solid
After test
Before test
No. 52—Gault brick, 4 inches thick, 4 inches solid

FIG. 70.—*Clay tile protections, test Nos. 52, 53, and 54, before and after fire test*

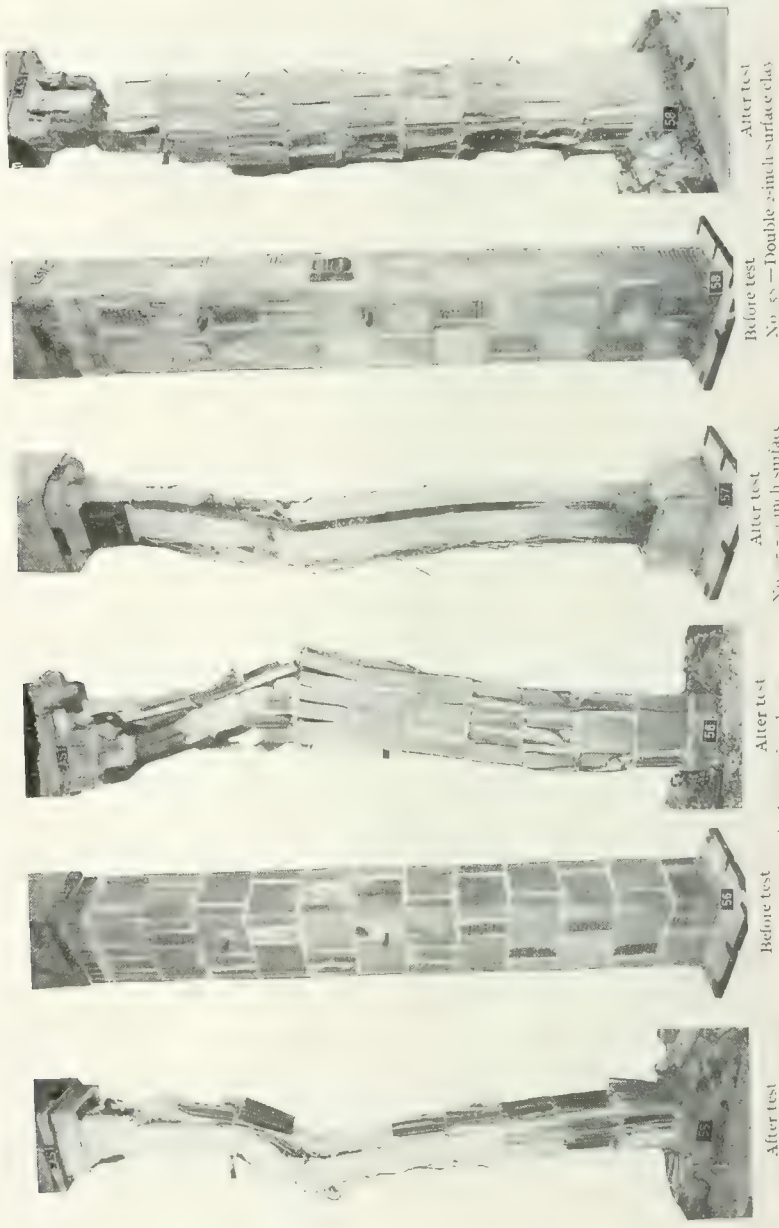
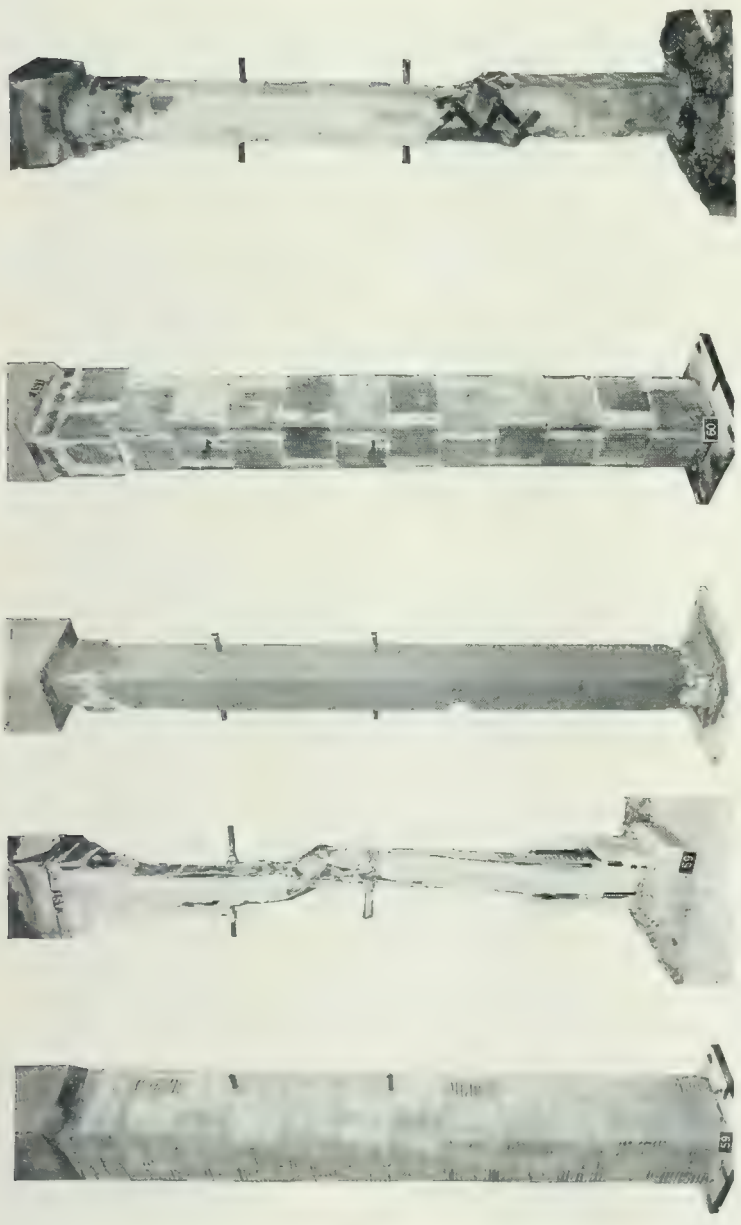


FIG. 74.—*Clay-tile protections, test Nos. 55 to 59, before and after fire test*



Before test. After test. After 4 hrs. After covering. After test.

No. 59. Double-mesh, surface clay. No. 60. Inch expanded clay.

FIG. 72.—4 hour-tile protection, Test Nos. 59 and 60, before and after fire test

FIG. 73.—Clay tile projections, test Nos. 61, 62, and 63, before and after the test

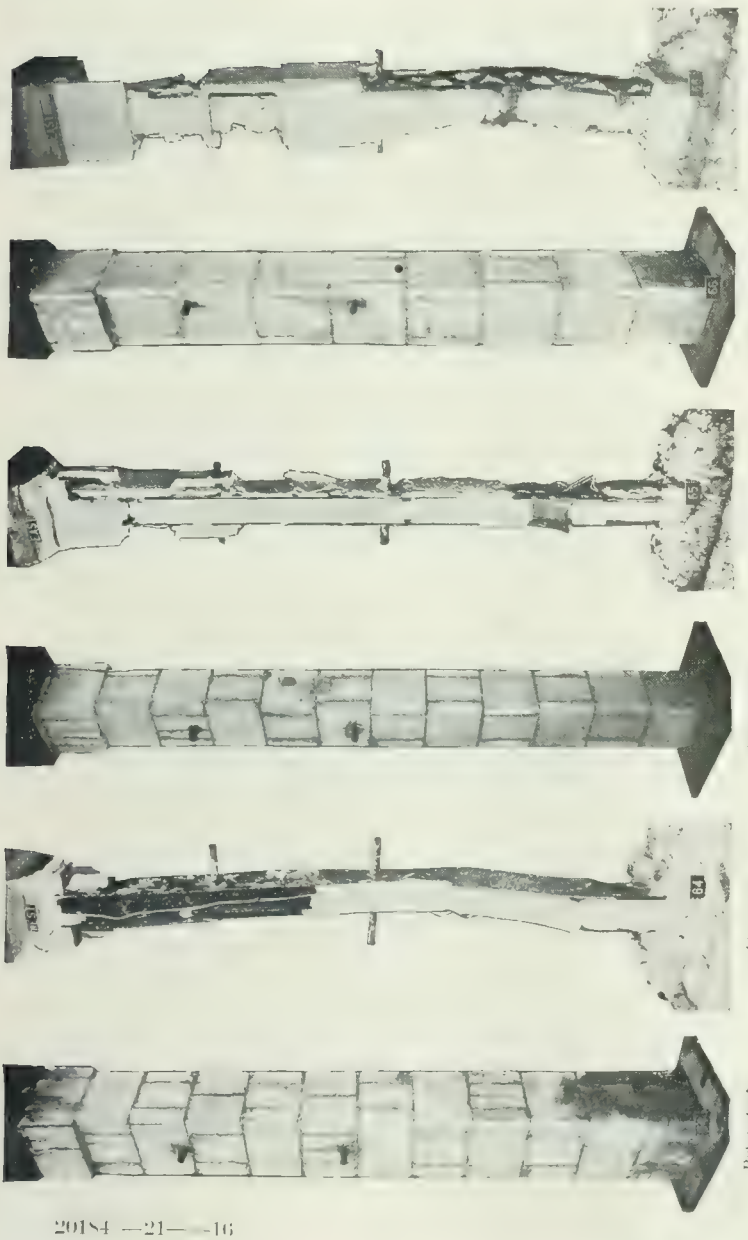


FIG. 74.—*Gypsum-block protections, test Nos. 64, 65, and 66, before and after fire test*

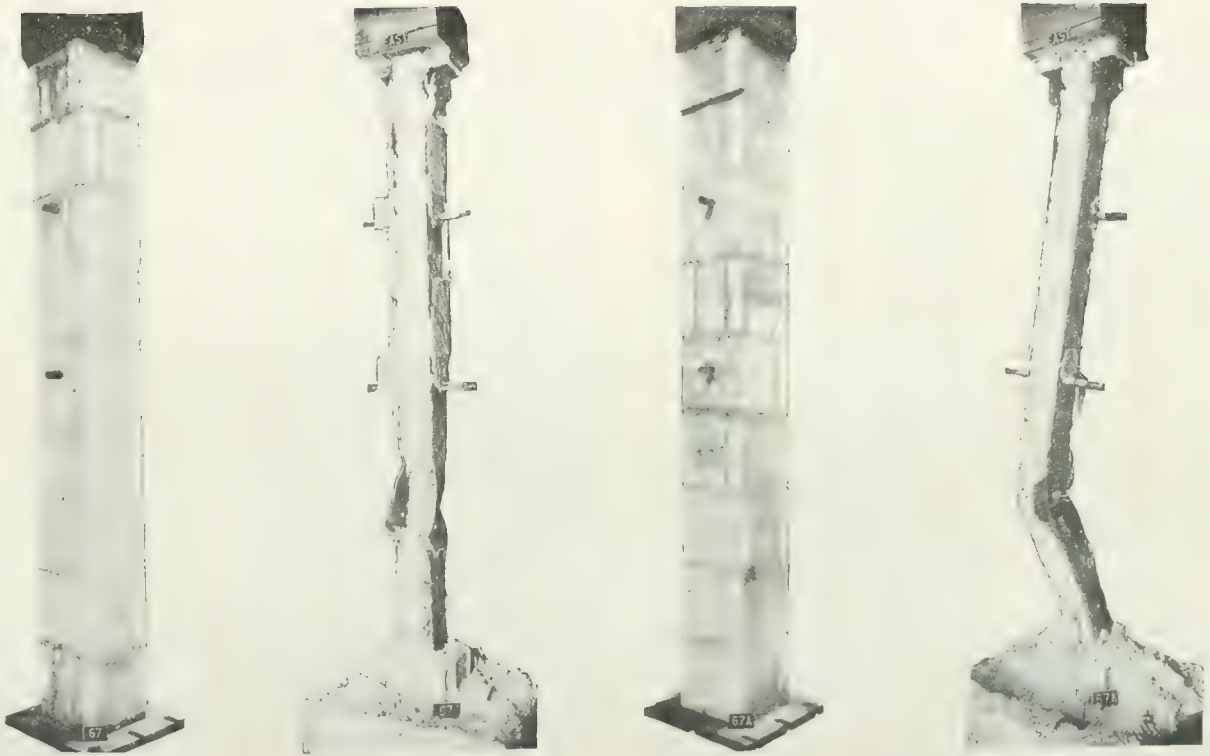
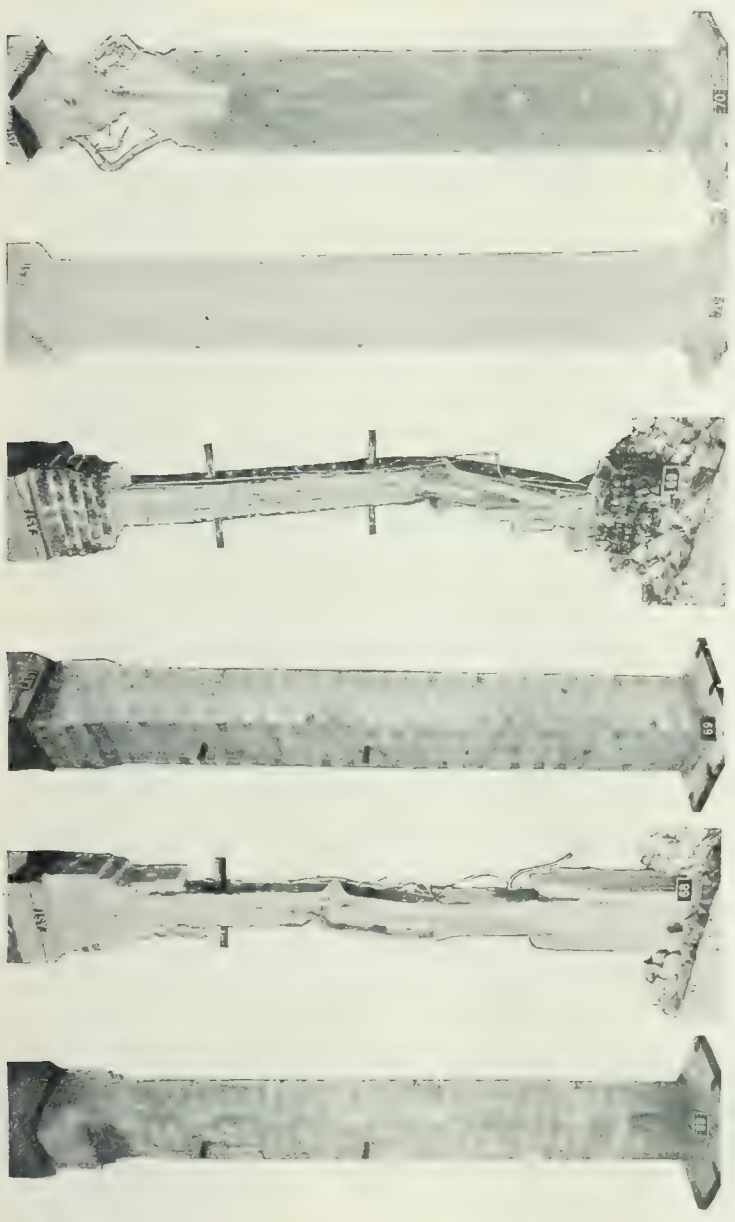


FIG. 75. *Gypsum block protections, test Nos. 67 and 67A, before and after fire test*



No. 68.—4-inch surface-clay brick No. 69.—3¼-inch surface-clay brick No. 70.—Vertically reinforced lime-stone concrete

FIG. 70.—*Brick protections, test Nos. 68 and 69; reinforced-concrete column, test No. 70, before and after fire test*

No. 71.—Vertically reinforced, trap-rock concrete

No. 72.—Vertically reinforced, limestone concrete

No. 73.—Vertically reinforced, trap-rock concrete

No. 74.—Hooped reinforced, limestone concrete

No. 75.—Hooped reinforced, trap-rock concrete

FIG. 77.—*Reinforced-concrete columns, test Nos. 71 to 75, after fire test*

FIG. 78. *Plastered clay-tile protections, test Nos. 76 and 77, before and after fire test*

FIG. 79.—*Longleaf pine column, protected, cap and pintle bearing, test No. 78, before and after fire test*

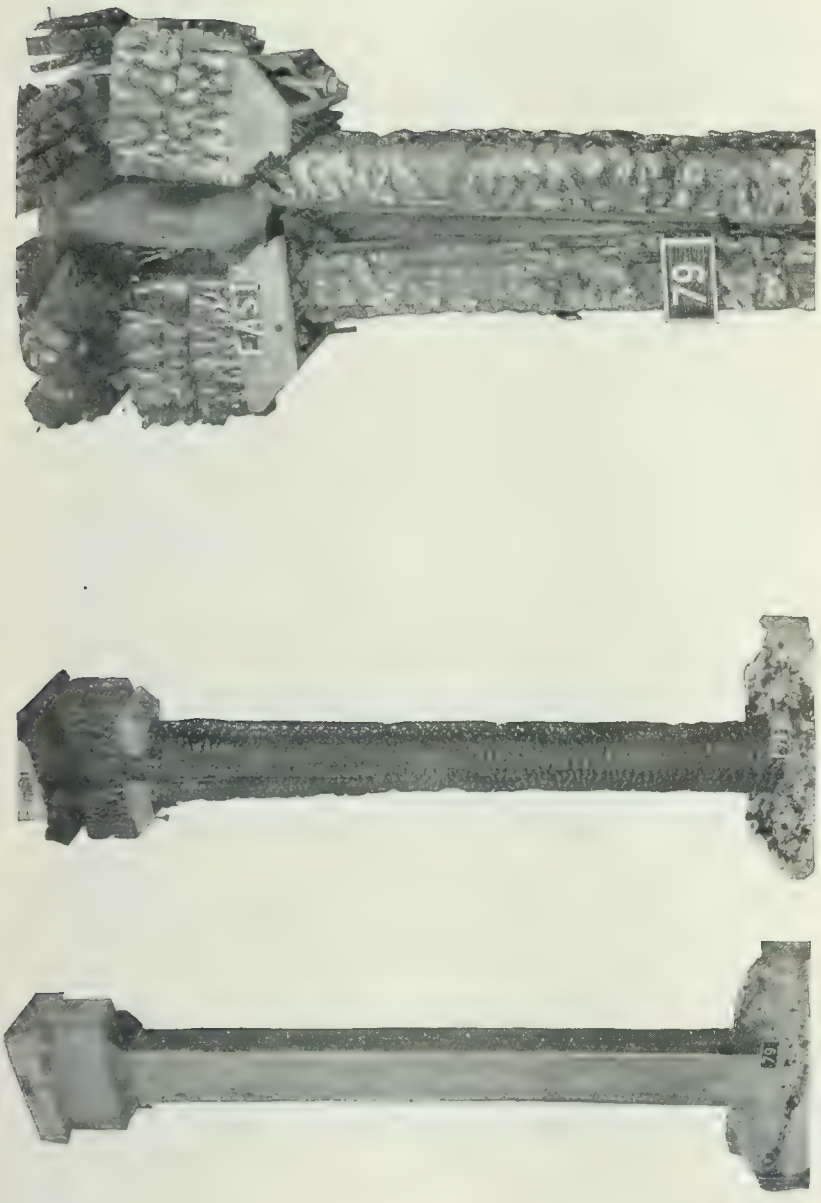
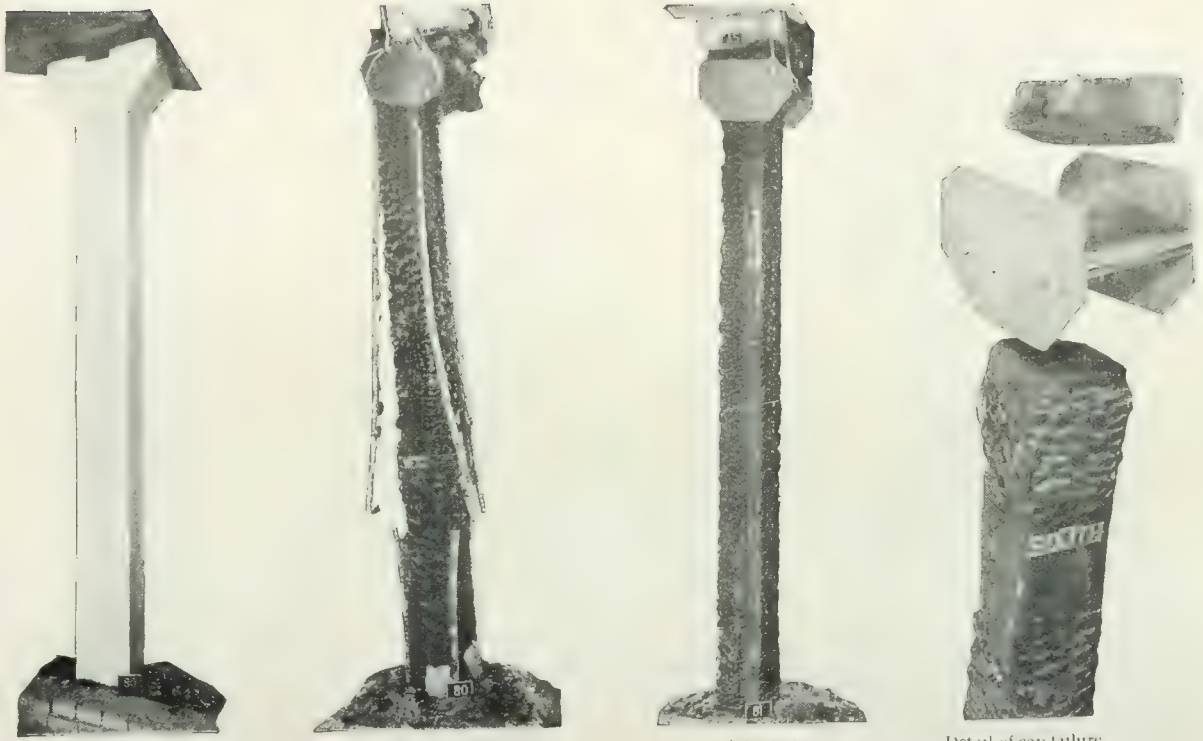


FIG. 80.—*Longleaf pine column, unprotected, cap and pintle bearing, test No. 79, before and after fire test*



Before test | After test | After test | Detail of cap failure

No. 80.—Gypsum wall-board protection | No. 81. Unprotected

FIG. 81.—*Longleaf pine columns, steel plate cap bearings, test Nos. 80 and 81, before and after fire test*

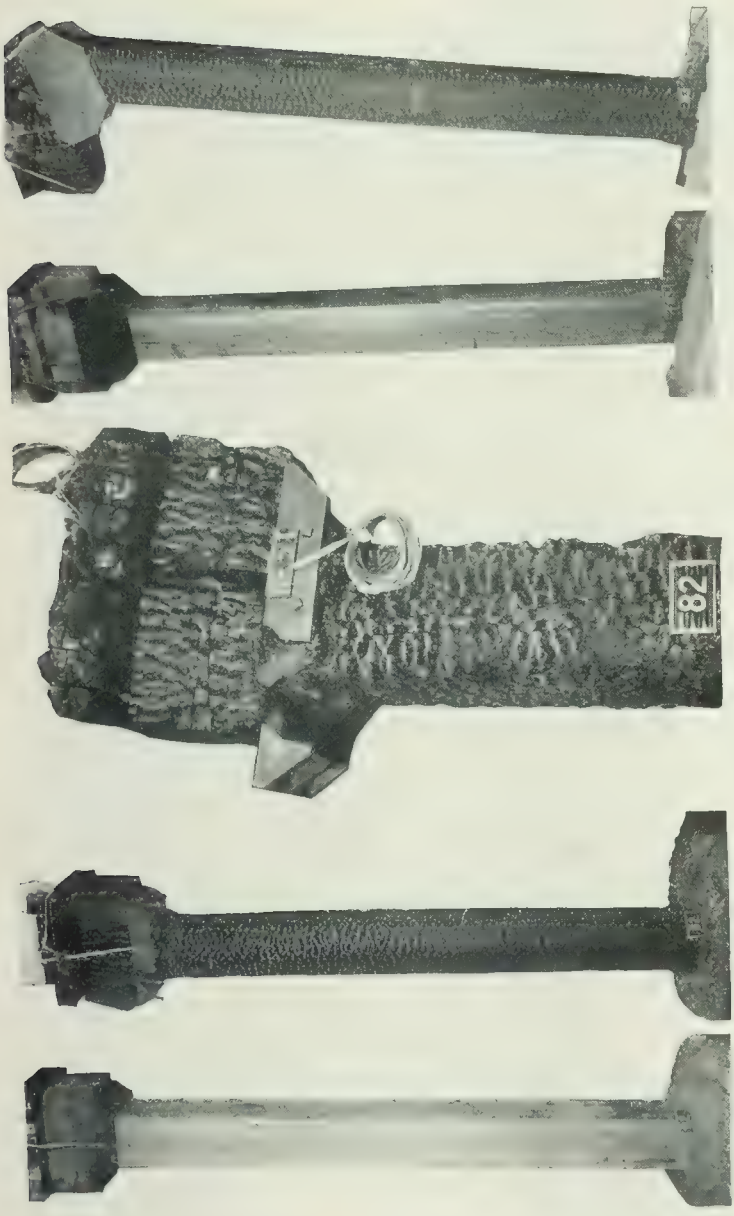


FIG. 82.—*Douglas fir columns, unprotected, test Nos. 82 and 83, before and after fire test*

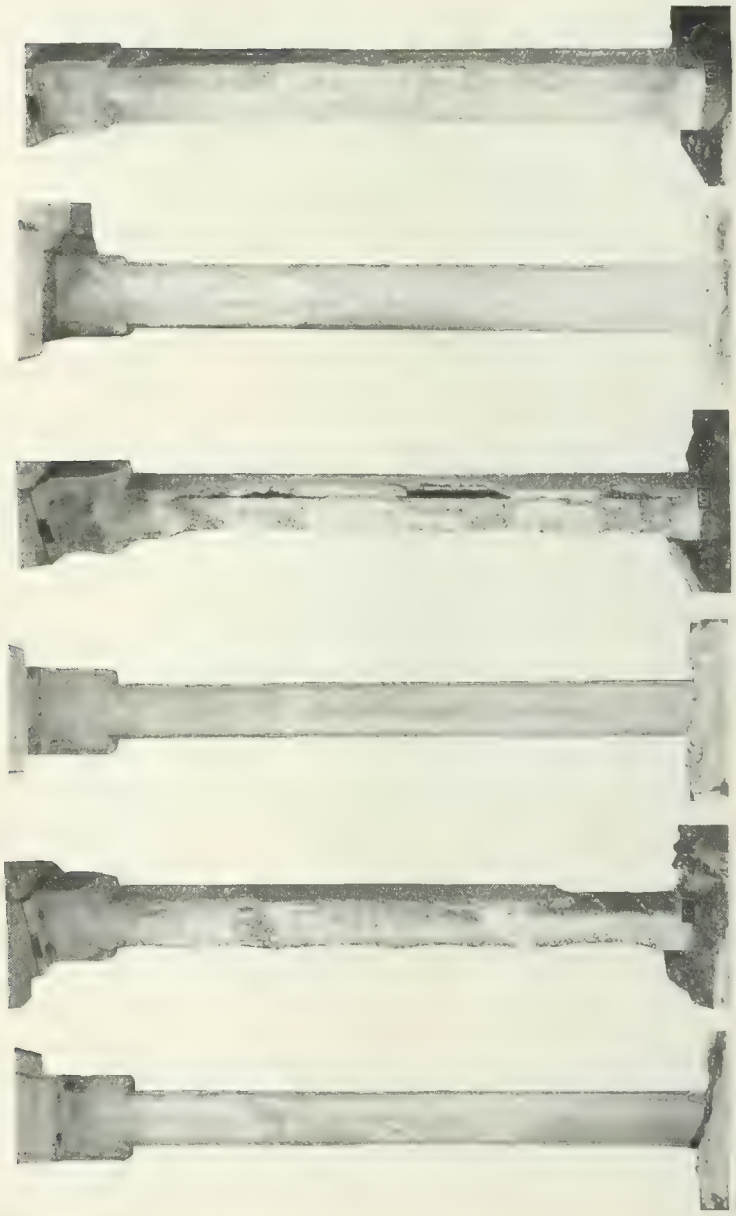


FIG. 83.—4-inch protections, test Nos. 103 to 105, after fire and salt test

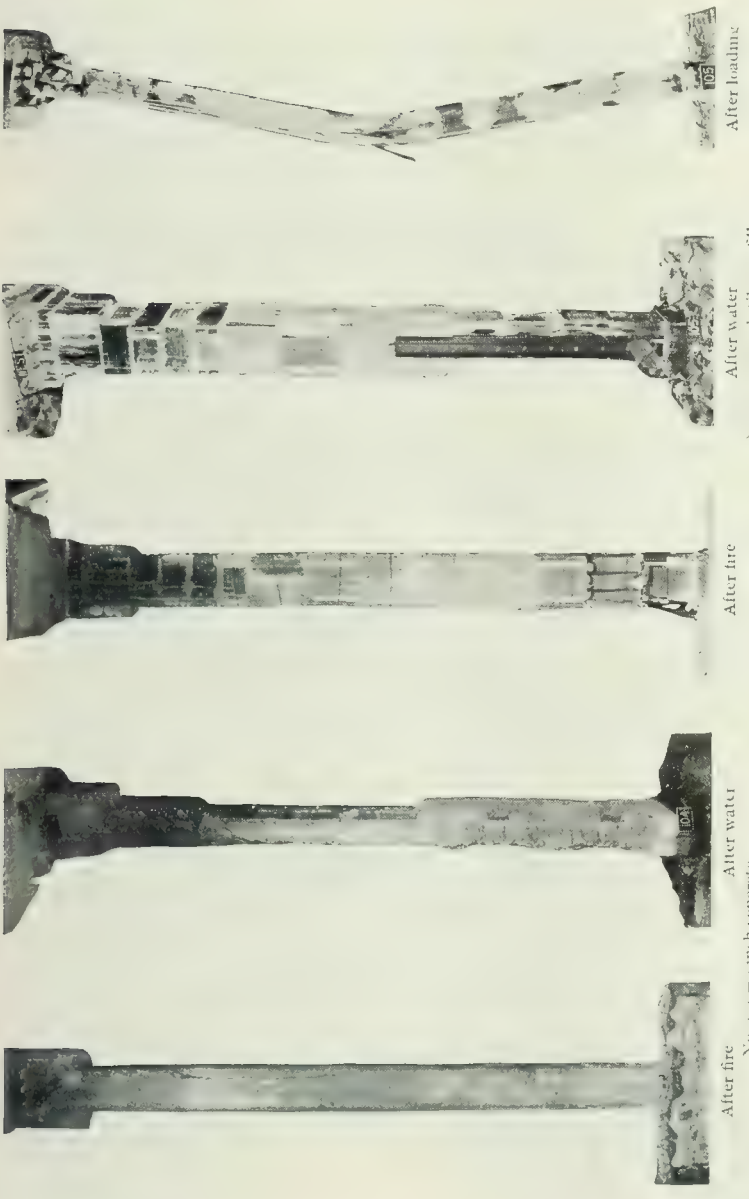


FIG. 84.—*Concrete protection, test No. 104; clay-tile protection, test No. 103, after fire and water tests*

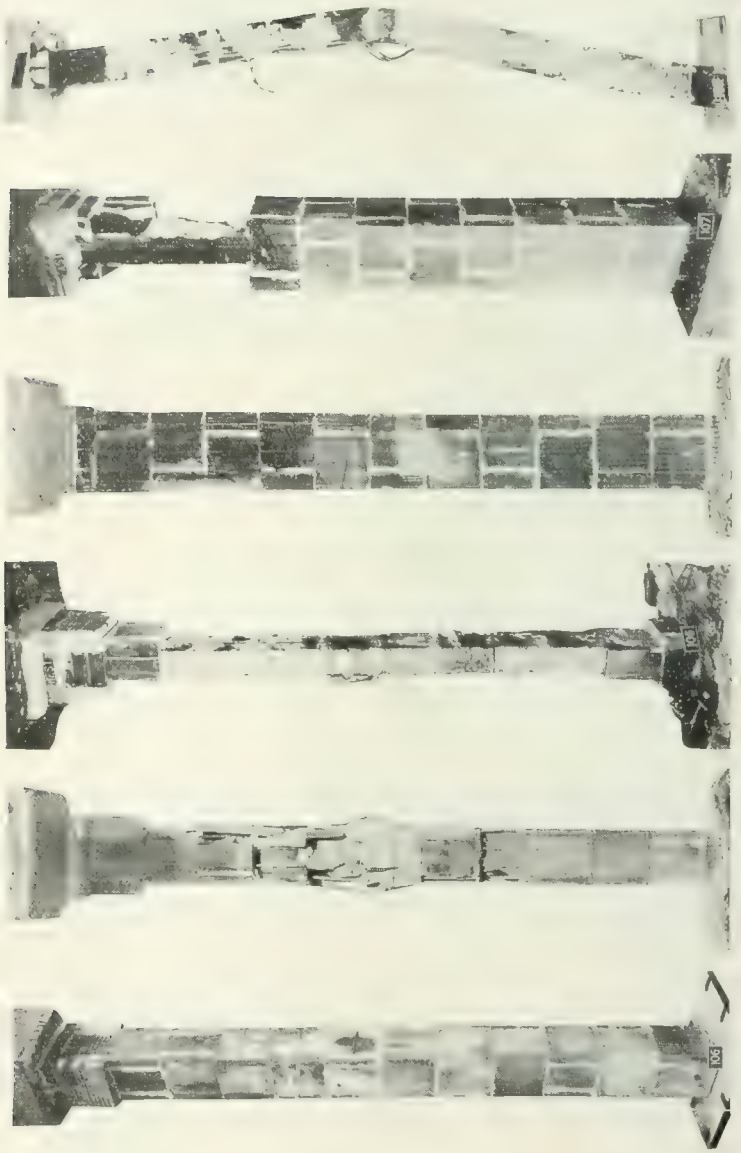


FIG. 85.—*Clay-tile protections test Nos. 126 and 127, after fire and water tests*

FIG. 80. *Gypsum block protections, test Nos. 108 and 109, after fire and water tests*

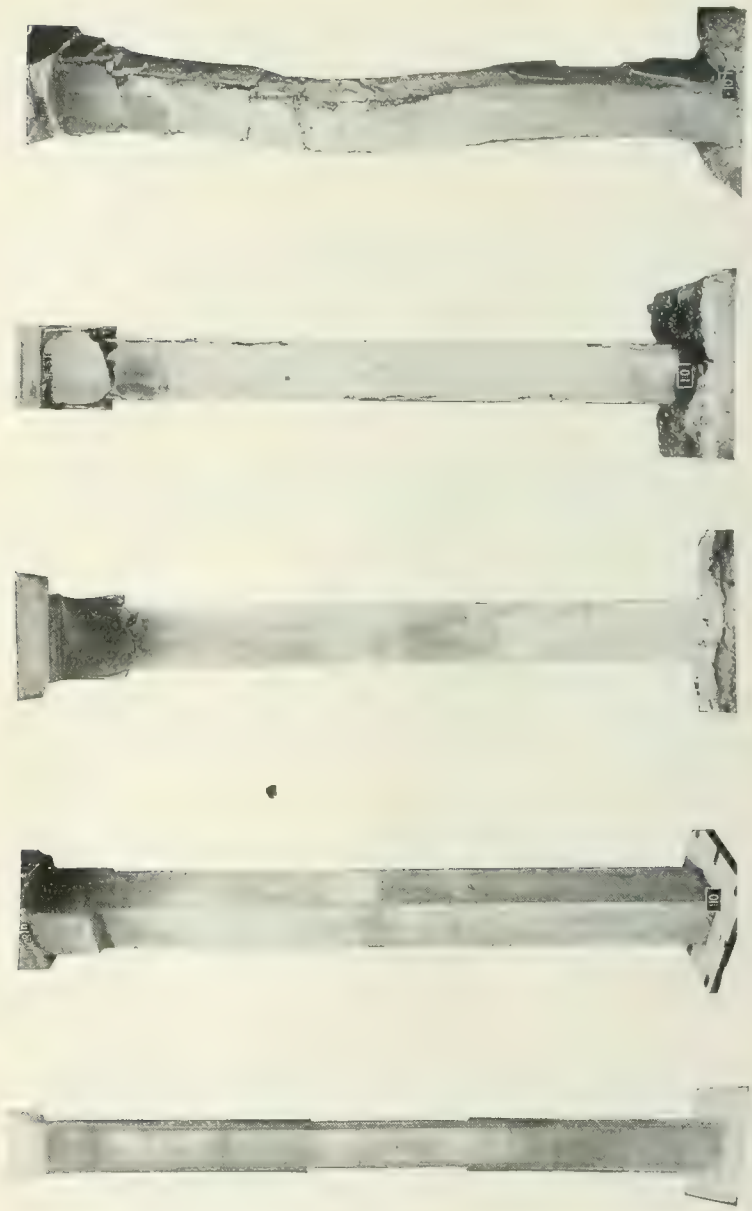


FIG. 87.—*Plaster on metal-lath protection, test No. 110, before and after fire and water tests*

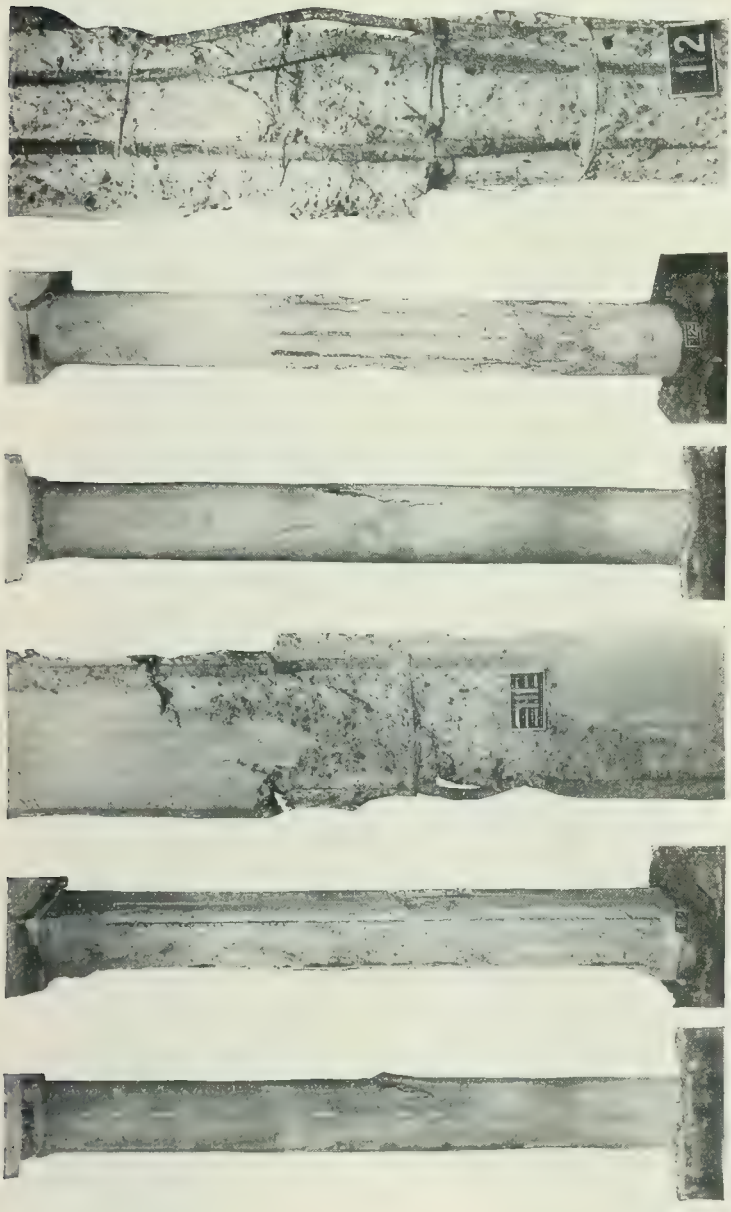


FIG. 88. *Reinforced concrete columns, tests Nos. 111 and 112, after fire and after tests*

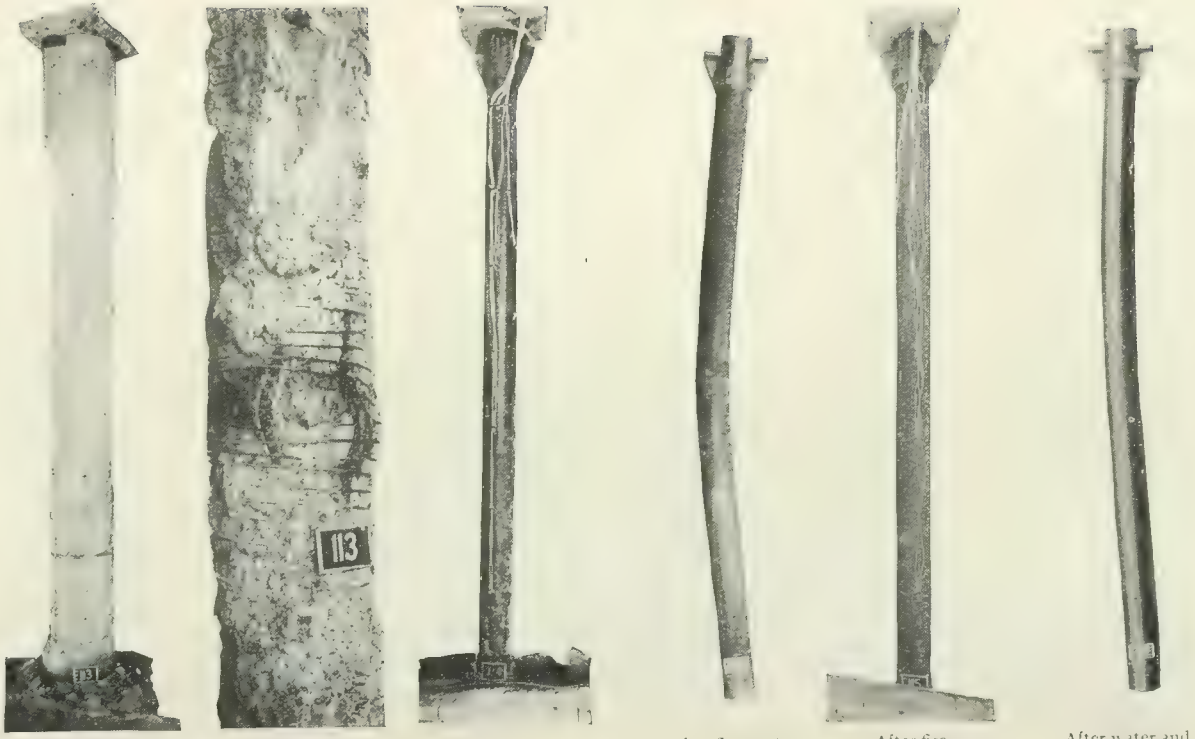


FIG. 89.—*Reinforced-concrete column, test No. 113; cast-iron columns, test Nos. 114 and 115, after fire and water tests*

APPENDIX B.—Time-Temperature Curves

Fig. No.	Test No.	Columns and protections	Page
90	1 to 4	Unprotected structural steel	251
91	5 to 8	....do	252
92	9 to 11	Unprotected cast iron	253
93	12, 13	Unprotected pipe	254
94	14 to 17	Partly protected structural steel	255
95	18, 22	....do	256
96	19, 20, 21	....do	257
97	23, 24	Double layer of plaster on metal lath	258
98	25, 26, 27	Single layer of plaster on metal lath	259
99	28	2-inch limestone concrete	260
100	28A	....do	261
101	29	2-inch trap-rock concrete	262
102	30	2-inch Joliet gravel concrete	263
103	31	2-inch sandstone concrete	264
104	32, 32A	2-inch cinder concrete	265
105	33	4-inch limestone concrete	266
106	33A	....do	267
107	34	4-inch granite concrete	268
108	34A	....do	269
109	35	4-inch limestone concrete	270
103	36	2-inch trap-rock concrete	264
110	37	4-inch trap-rock concrete	271
111	38	2-inch Joliet gravel concrete	272
112	39	4-inch Meramec River gravel concrete	273
113	40	2-inch trap-rock concrete	274
114	41	4-inch limestone concrete	275
115	42	....do	276
116	43, 44	2-inch sandstone concrete	277
112	45	2-inch Meramec River gravel concrete	273
117	46	2-inch trap-rock concrete (3⅛ inch outside of angles)	278
101	47	2-inch cinder concrete	262
118	48, 49	Semifire-clay tile	279
119	50, 50A	Surface-clay tile	280
120	51, 51A	....do	281
121	52, 53	Shale tile	282
122	54, 55	Semifire-clay tile	283
123	56	Semifire clay	284
123	57	Surface-clay tile	284
124	58, 59	....do	285
125	60, 61	Semifire-clay tile	286
126	62, 63	Porous semifire-clay tile	287
127	64	4-inch gypsum block	288
128	65, 66	2-inch gypsum block	289
129	67	4-inch gypsum block	290
130	67A	....do	291
118	68	2¼-inch surface-clay brick	279
131	69	3¾-inch surface-clay brick	292
132	70	Square, reinforced, limestone concrete	293
133	71	Square, reinforced, trap-rock concrete	294
134	72	Round, reinforced, limestone concrete	295

APPENDIX B.—Time-Temperature Curves—Continued

Fig. No.	Test No.	Columns and protections	Page
135	73	Round, reinforced, trap-rock concrete	296
136	74	Round, hooped, limestone concrete	297
137	75	Round, hooped, trap-rock concrete	298
138	76	Plastered-clay tile	299
139	77	do	300
140	78, 79, 82	Timber columns, cap and pintle	301
141	80, 81, 83	Timber columns, steel-plate cap	302
142	101 to 104	Concrete, fire and water tests	303
143	105 to 107	Clay tile, fire and water tests	304
143	108, 109	Gypsum block, fire and water tests	304
144	110	Plaster on metal lath, fire and water tests	305
145	111 to 113	Reinforced concrete, fire and water tests	306
145	114, 115	Unprotected cast iron, fire and water tests	306

NOTES

The upper curves in the diagrams, usually four in number, give indicated furnace temperatures at four points within the furnace chamber, two at 3 feet and two at 9 feet above the fireproofing line at the base of the test column, the lower location being designated by *L* and the upper *U*. The other letters of the curve designations indicate the corner of the furnace nearest to the given thermocouple location. Thus:

L–NW, lower northwest location
L–SE, lower southeast location
U–NE, upper northeast location
U–SW, upper southwest location

cf. Section VII, paragraph *2a*, and Fig. 31.

The lower curves in the diagrams give temperatures at four general levels on the test column, and for one or two of the levels the temperatures are given for several points on the section, as shown by the location diagram given for each test. The levels are indicated by the letters *L*, *N*, *M*, and *T*, for locations 1½, 4½, 7½, and 10½ feet, respectively, above the fireproofing line at the base of the column. Thus for test No. 14, Fig. 94:

B2, 1½-foot level at center of column flange
N1, 4½-foot level at center of column web
N2, 4½-foot level at center of column flange
M1, 7½-foot level at center of column web
M2, 7½-foot level at center of column flange
M3, 7½-foot level at edge of column flange
T2, 10½-foot level at center of column flange

cf. Section VII, paragraph *3b*, and Fig. 31.

The arrows in Figs. 90 to 141 indicate the time of failure of the columns in the fire tests. The arrows in Figs. 142 to 145 indicate the time water was applied to the columns in the fire and water tests.

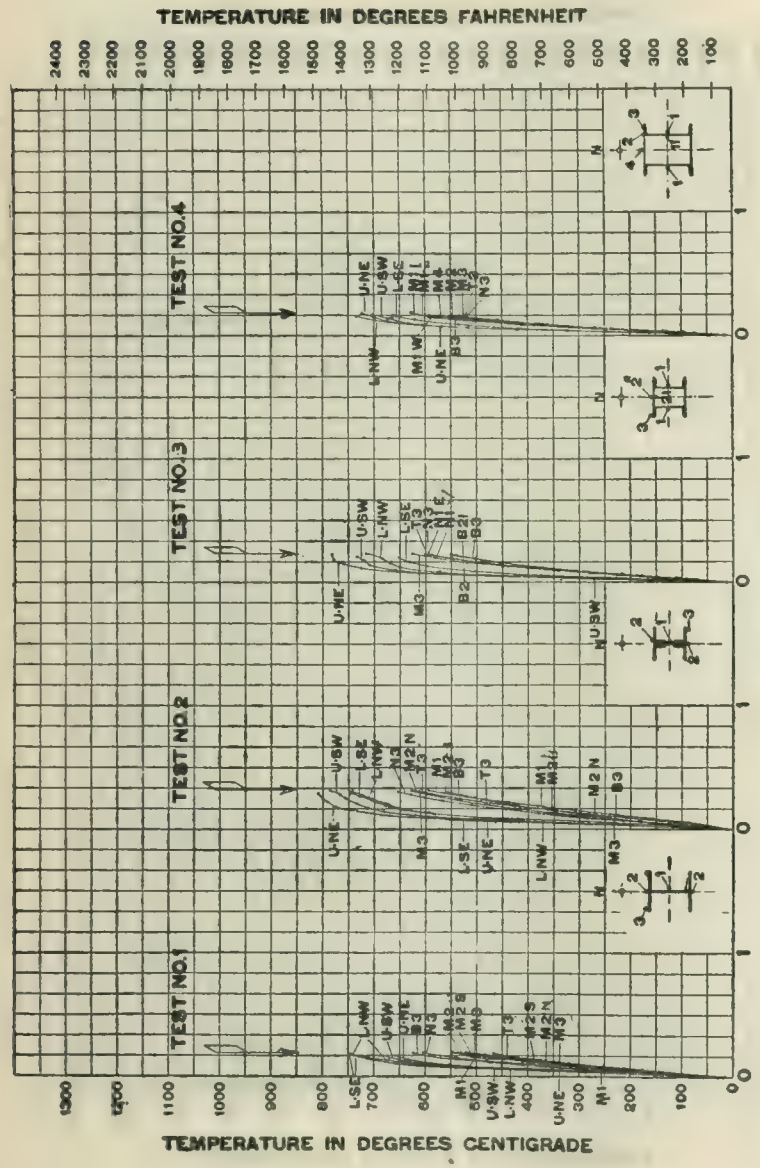


FIG. 90.—*Furnace and column temperatures, unprotected structural-steel columns, test Nos. 1 to 4*

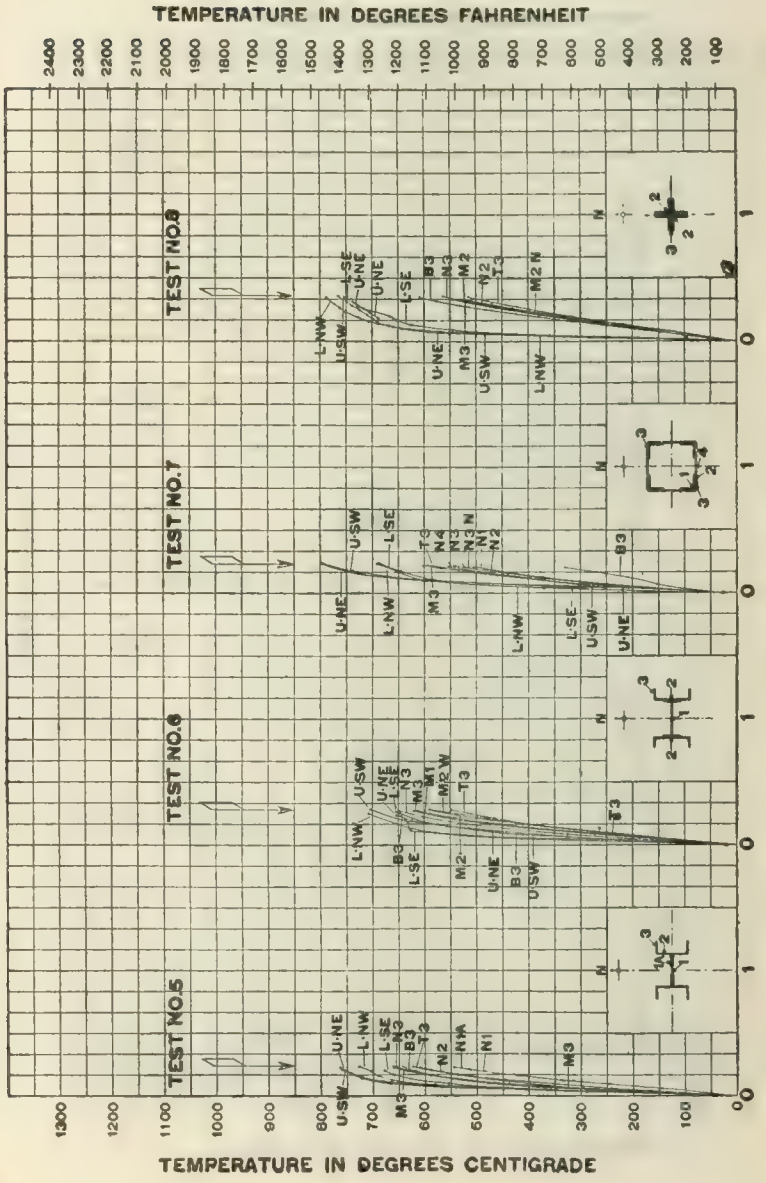


FIG. 91.—*Furnace and column temperatures, unprotected structural-steel columns, test Nos. 5 to 8*

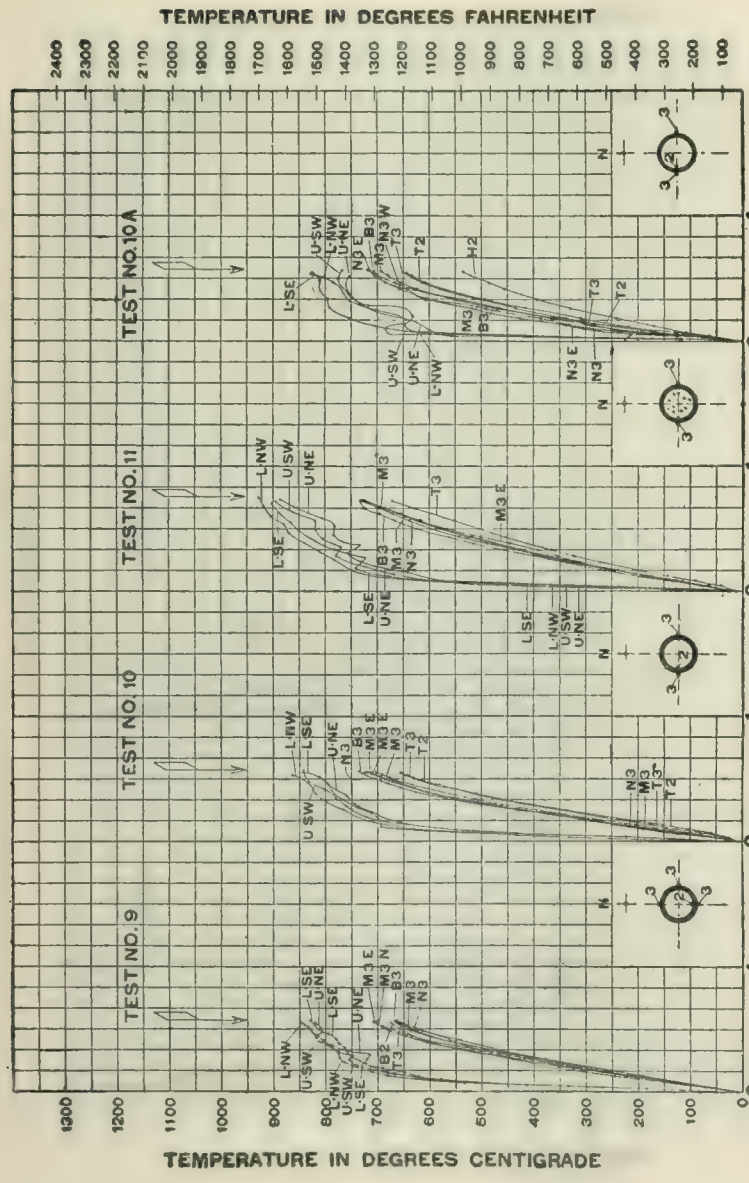


FIG. 92.—*Furnace and column temperatures, unprotected cast-iron columns, test Nos. 9 to 11*

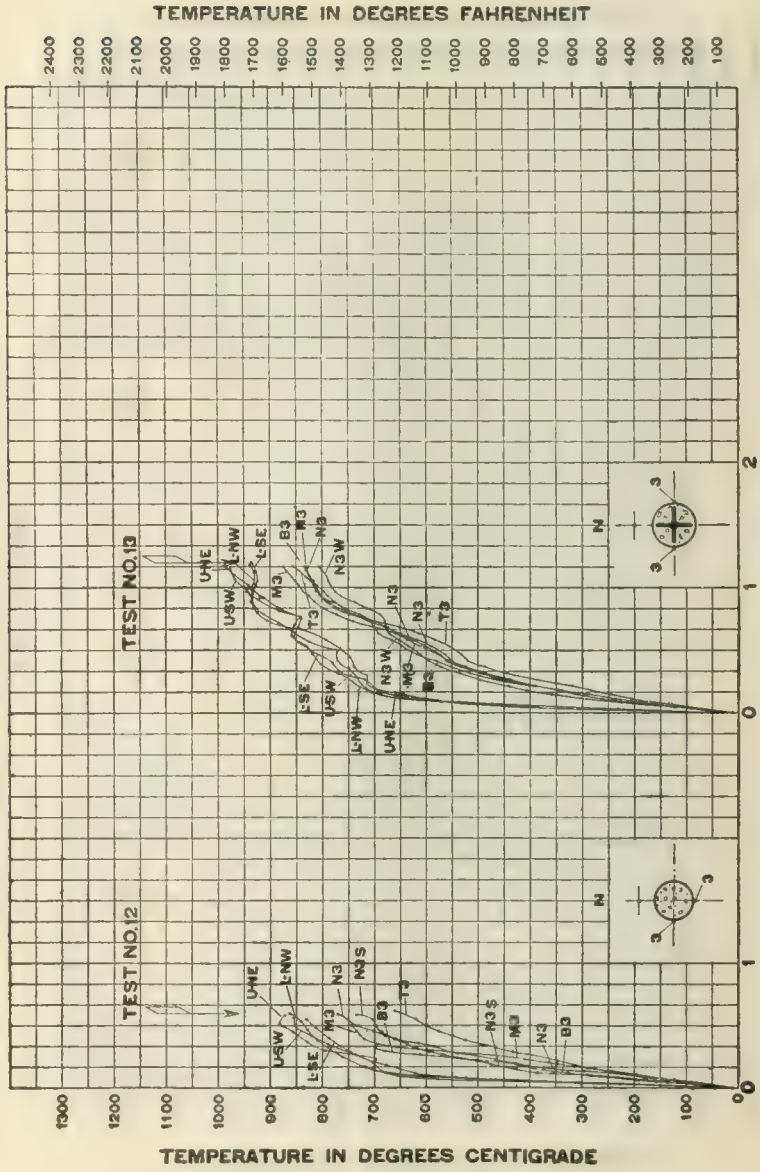


FIG. 93.—*Furnace and column temperatures, unprotected pipe columns, test Nos. 12 and 13*

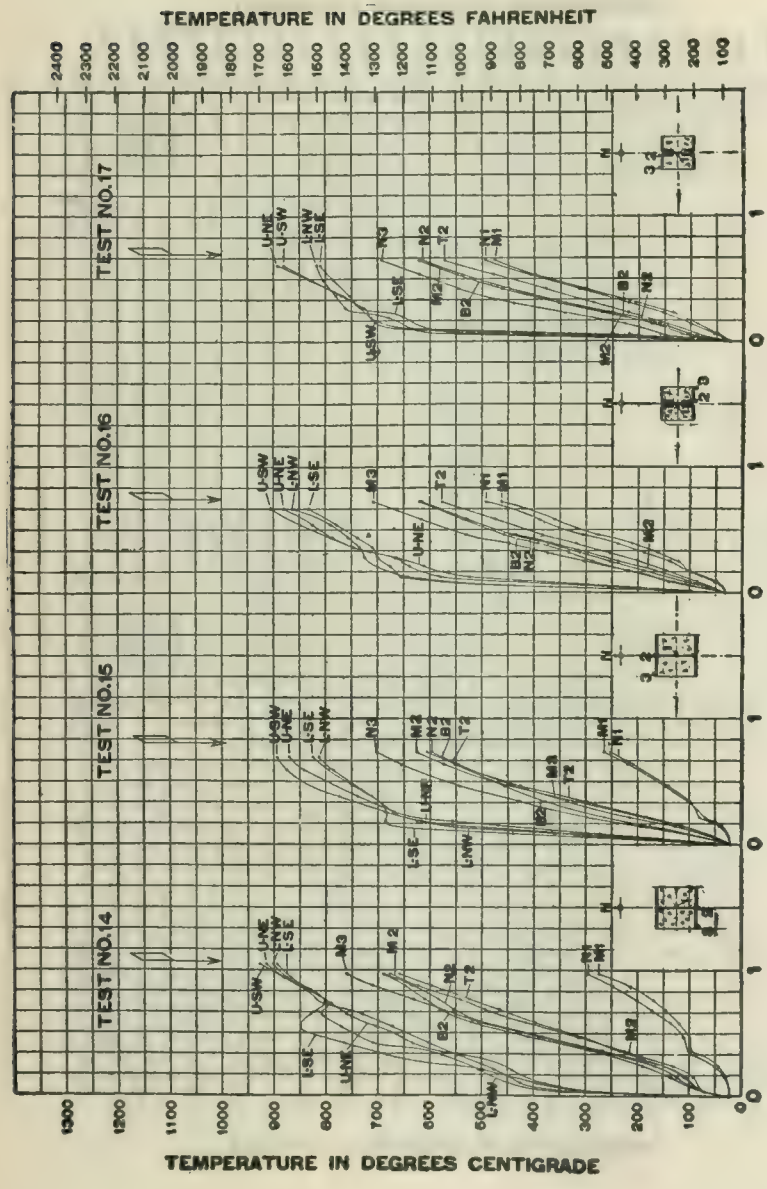


FIG. 94.—*Furnace and column temperatures, partly protected structural-steel columns, test Nos. 14 to 17*

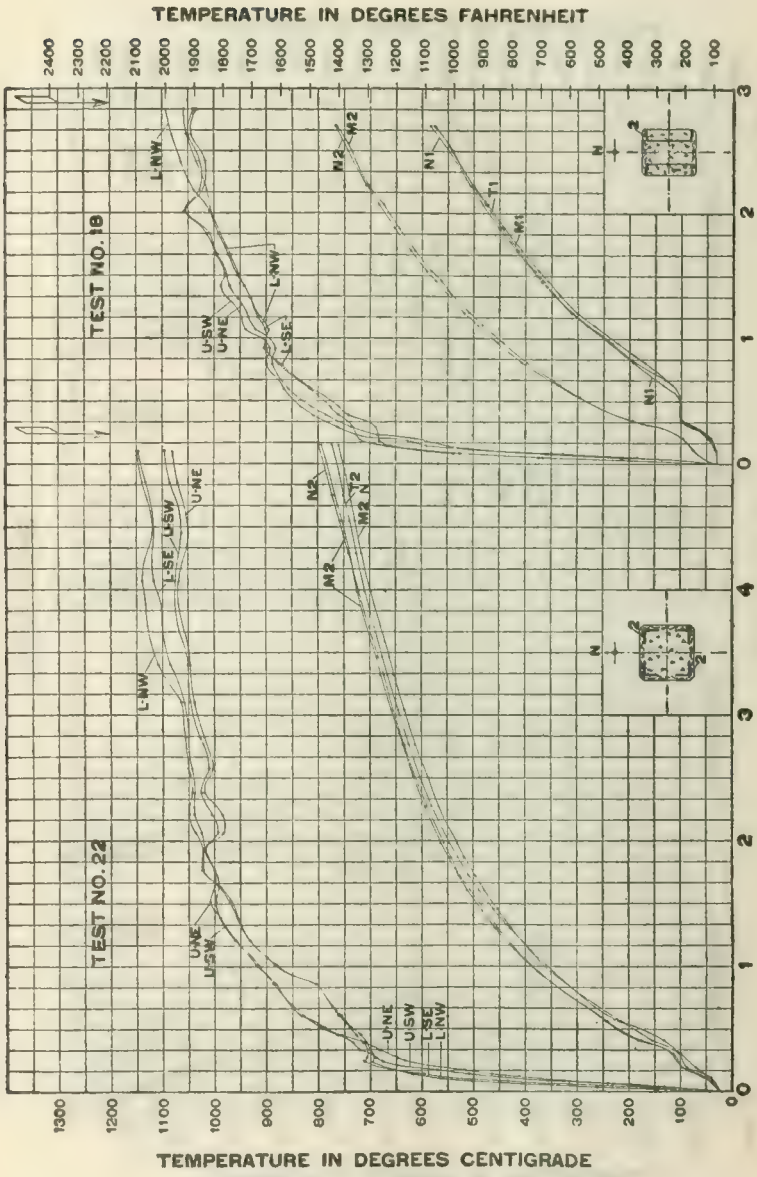


FIG. 95.—*Furnace and column temperatures, partly protected structural-steel columns, test Nos. 18 and 22*

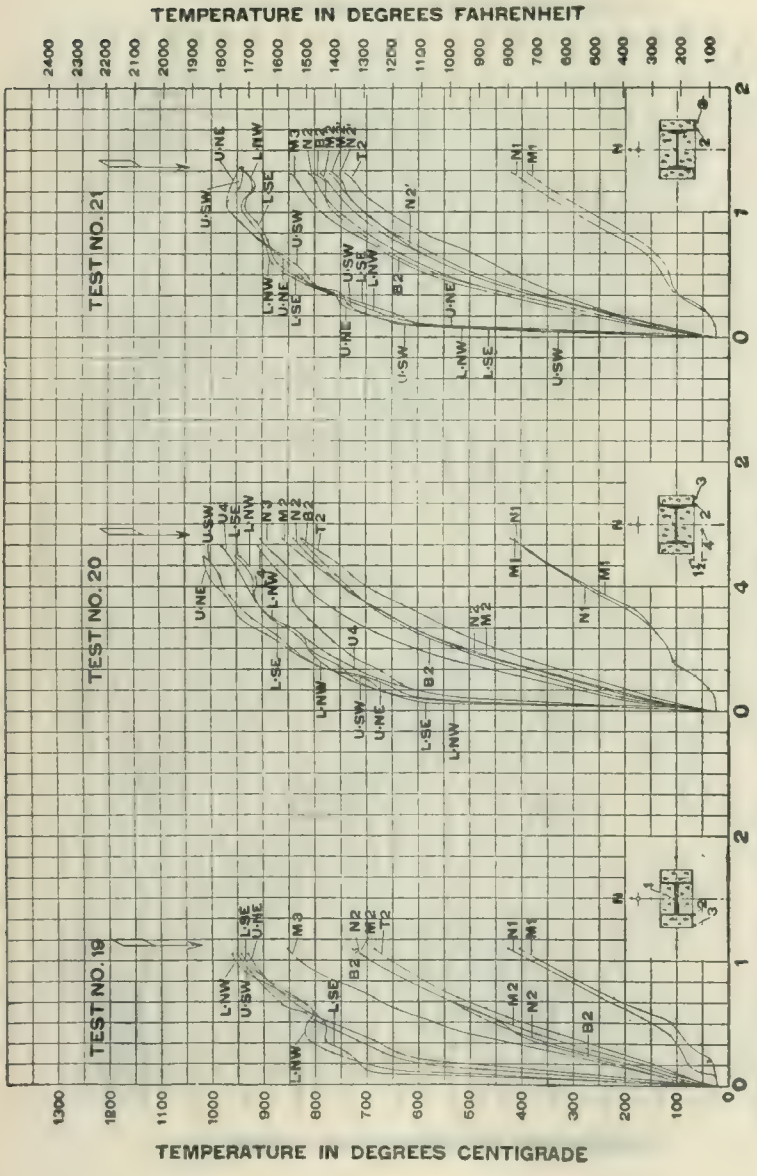


FIG. 96.—*Furnace and column temperatures, partly protected structural-steel columns, test Nos. 19, 20, and 21*

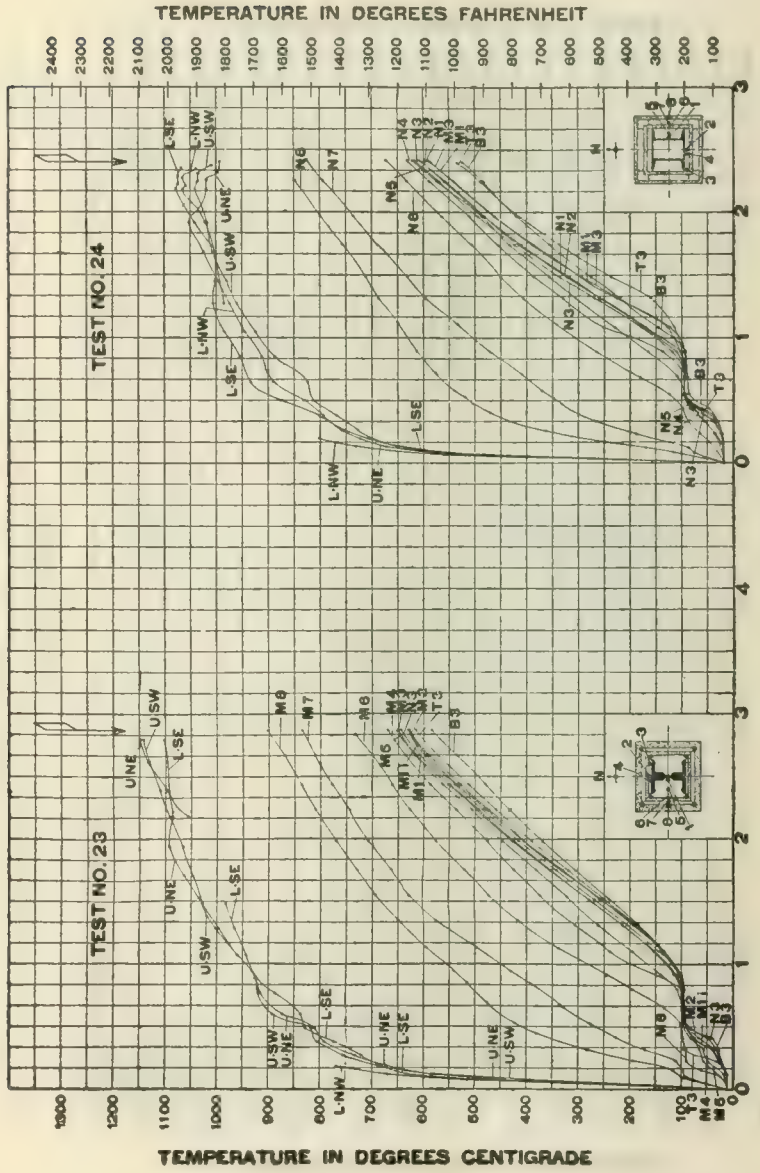


FIG. 97.—*Furnace and column temperatures, plaster on metal-lath protections, test Nos. 23 and 24*

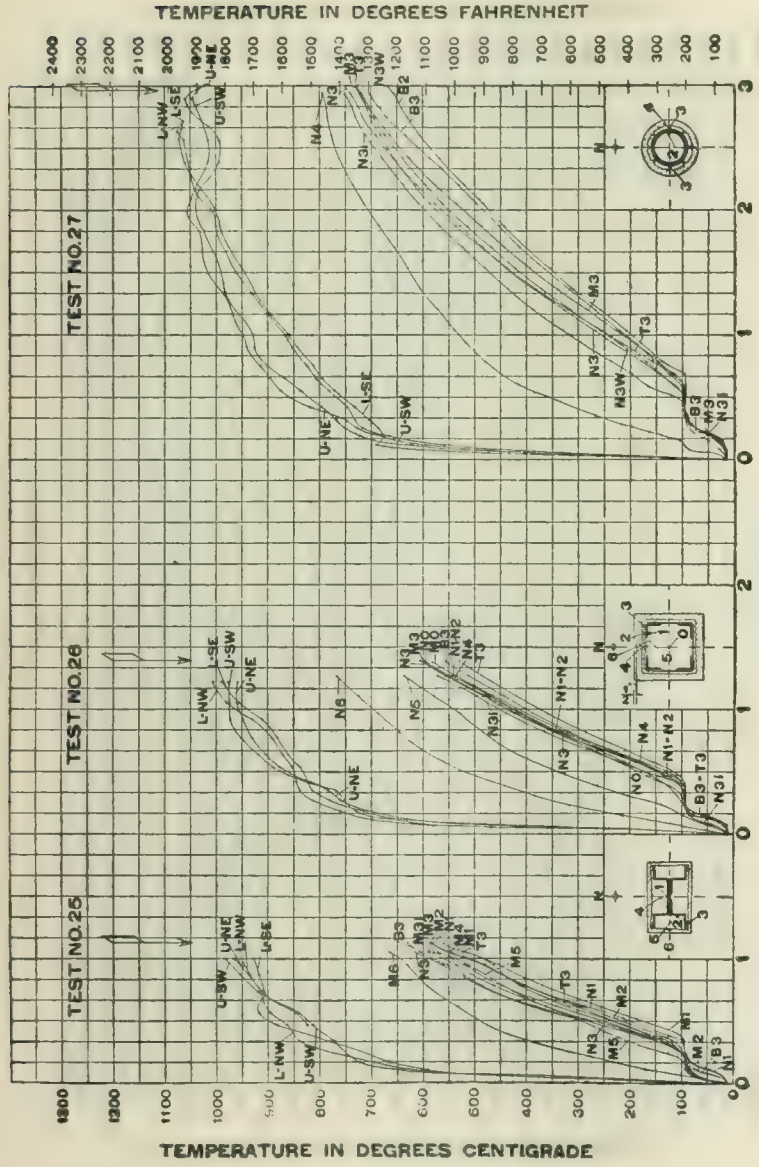


FIG. 98.—*Furnace and column temperatures, plaster on metal-lath protections, test Nos. 25, 26, and 27*

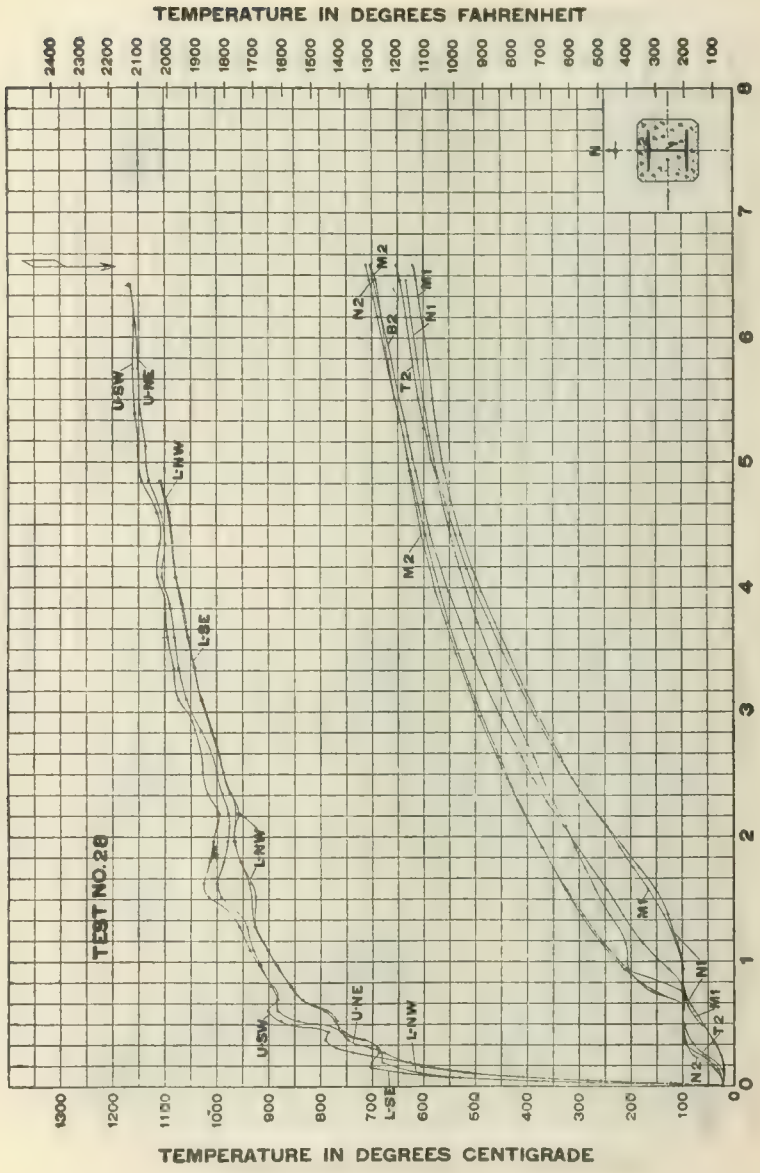


FIG. 99.—*Furnace and column temperatures, 2-inch limestone concrete protection, test No. 28*

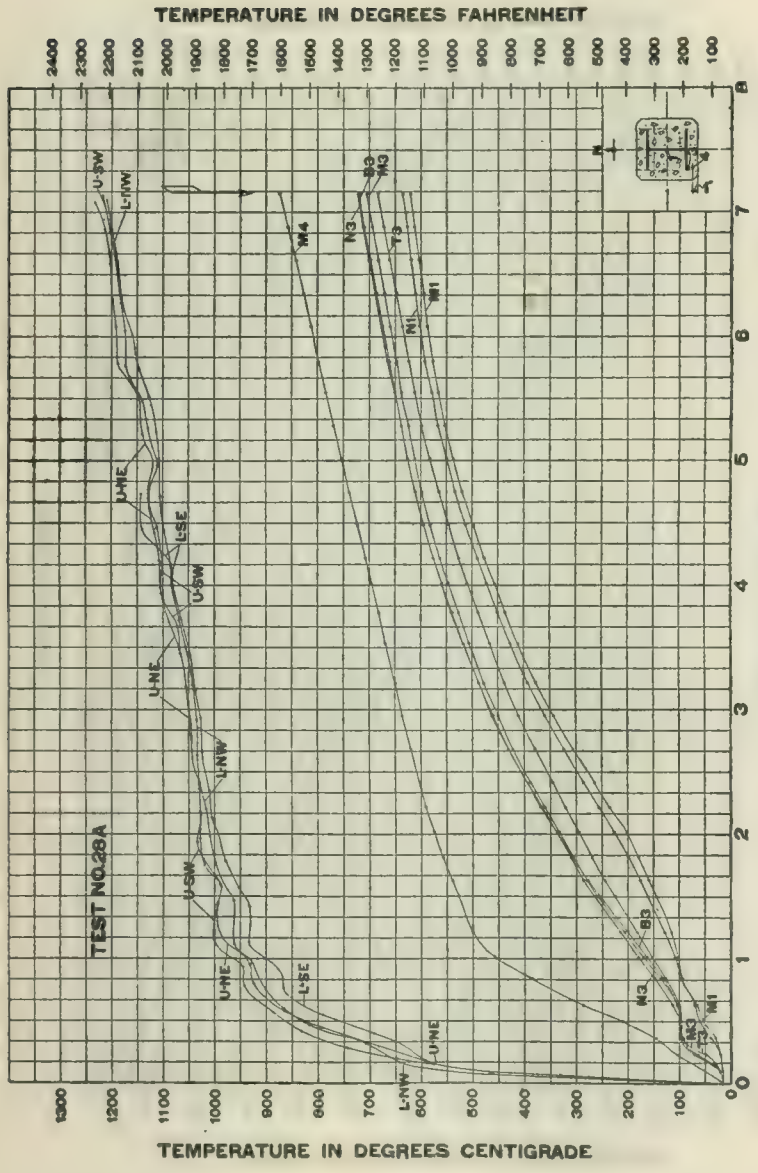


FIG. 100.—*Furnace and column temperatures 2-inch limestone concrete protection, test No. 28A*

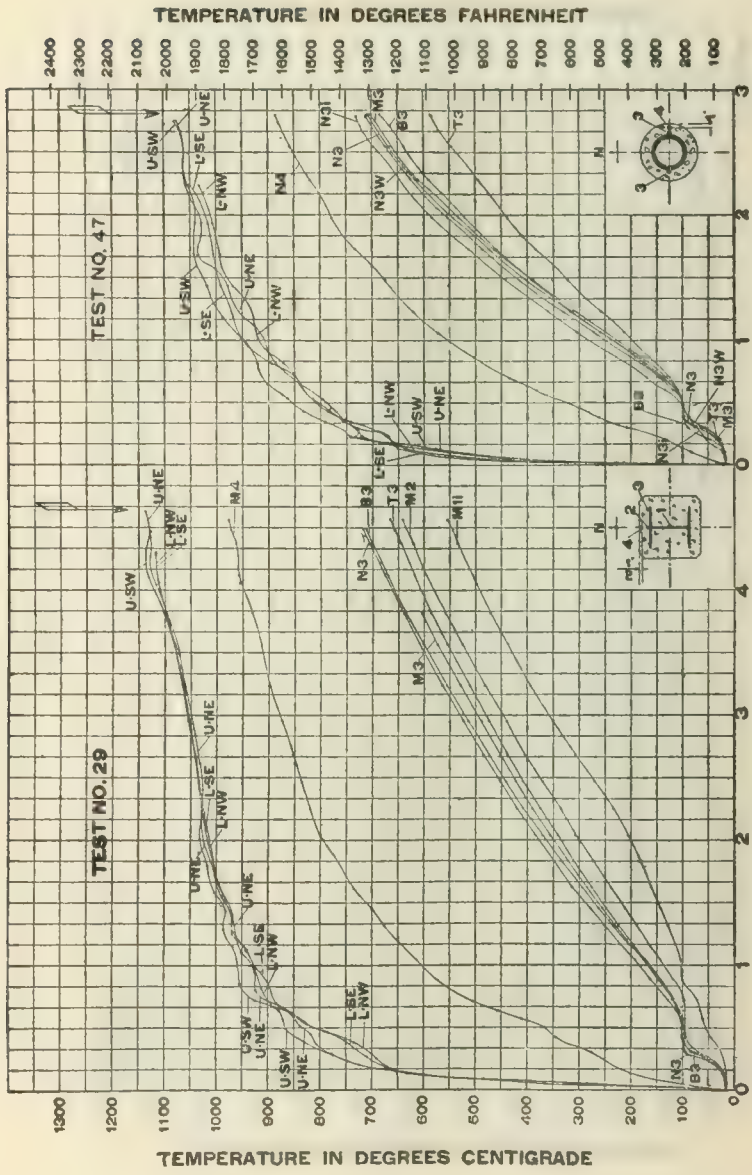


FIG. 101.—*Furnace and column temperatures, 2-inch trap-rock concrete protection, test No. 29; 2-inch cinder concrete protection, test No. 47*

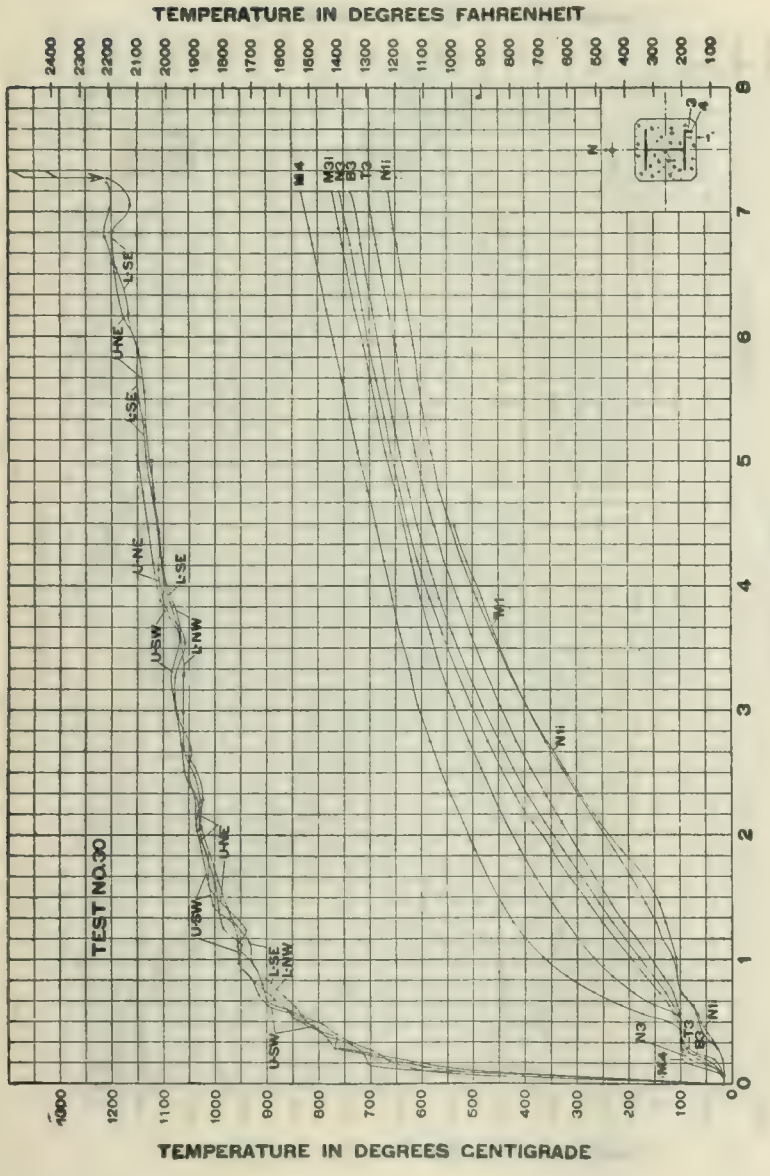


FIG. 102.—*Furnace and column temperatures, 2-inch Joliet gravel concrete protection, test No. 30*

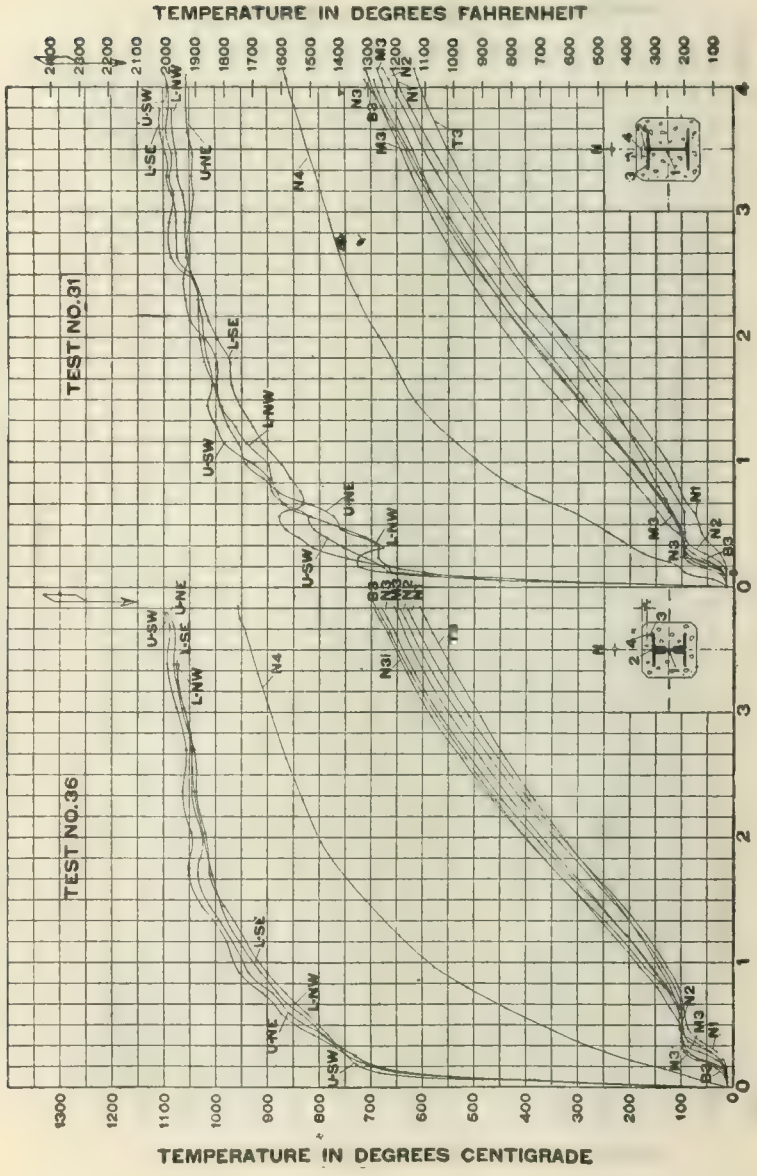


FIG. 103.—*Furnace and column temperatures, 2-inch sandstone concrete protection, test No. 31; 2-inch trap-rock concrete protection, test No. 36*

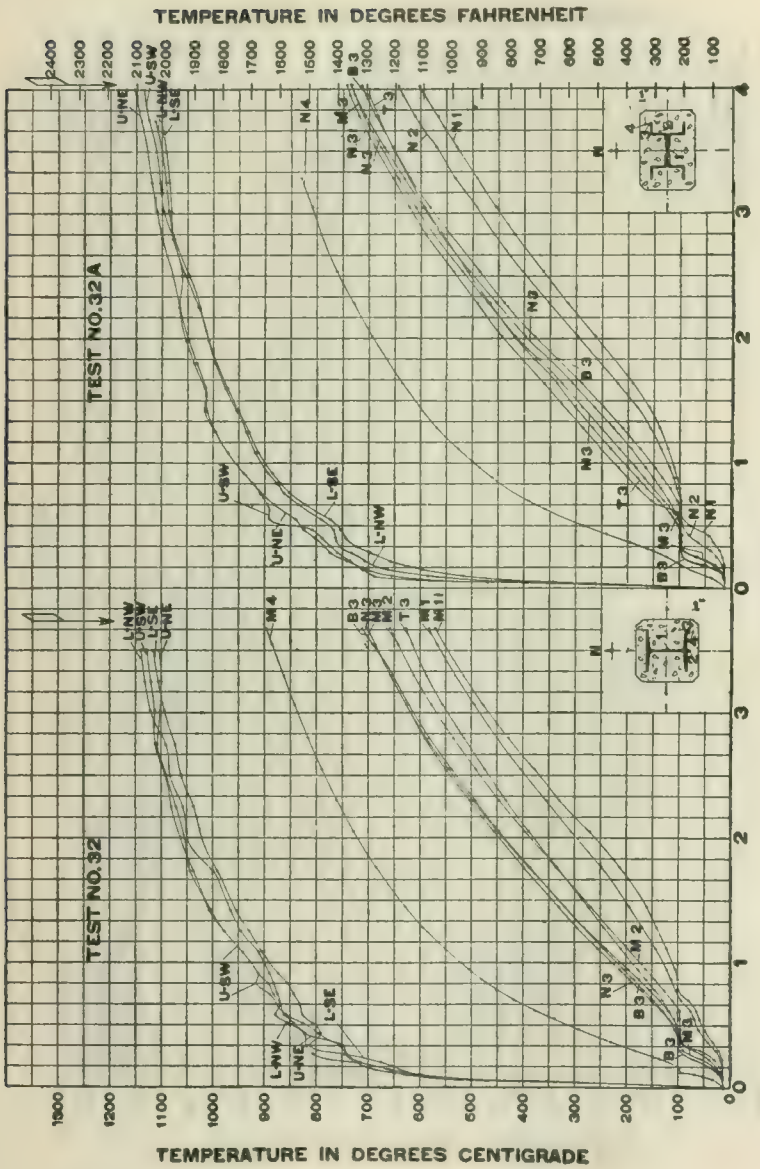


FIG. 104.—*Furnace and column temperatures, 2-inch cinder concrete protections, test Nos. 32 and 32A*

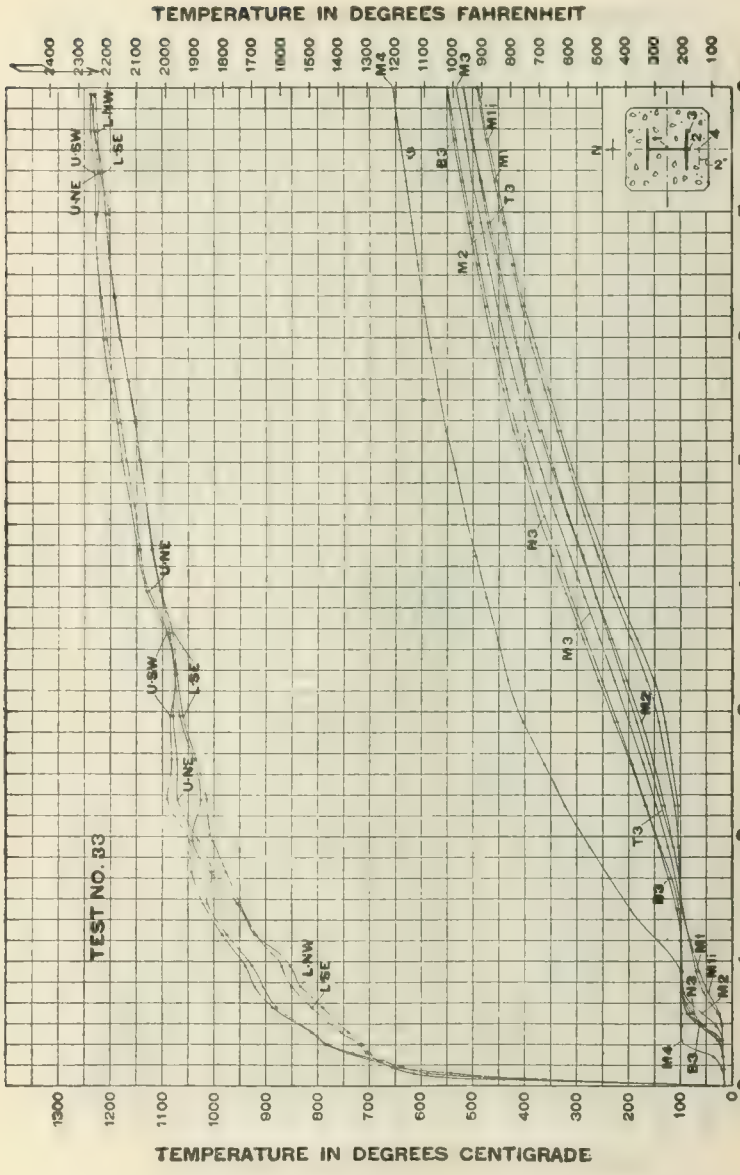


FIG. 105.—*Furnace and column temperatures, 4-inch limestone concrete protection, test No. 33*

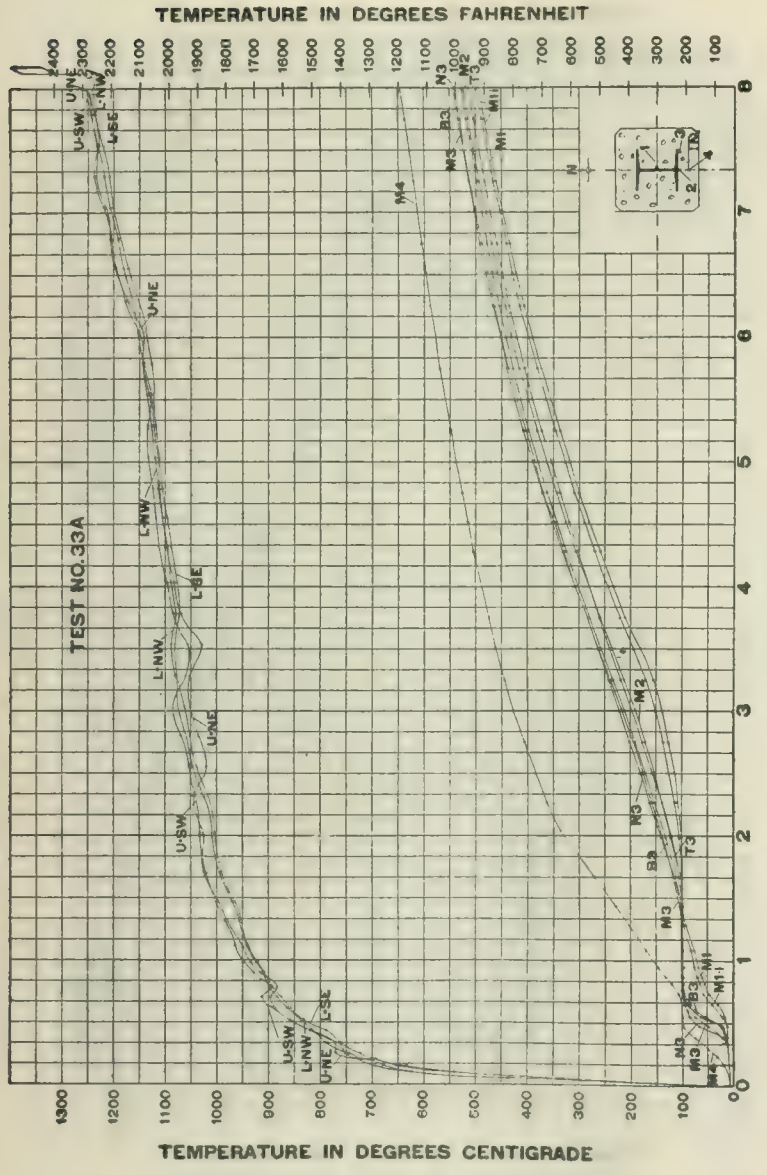


FIG. 106.—*Furnace and column temperatures, 4-inch limestone concrete protection, test No. 33A*

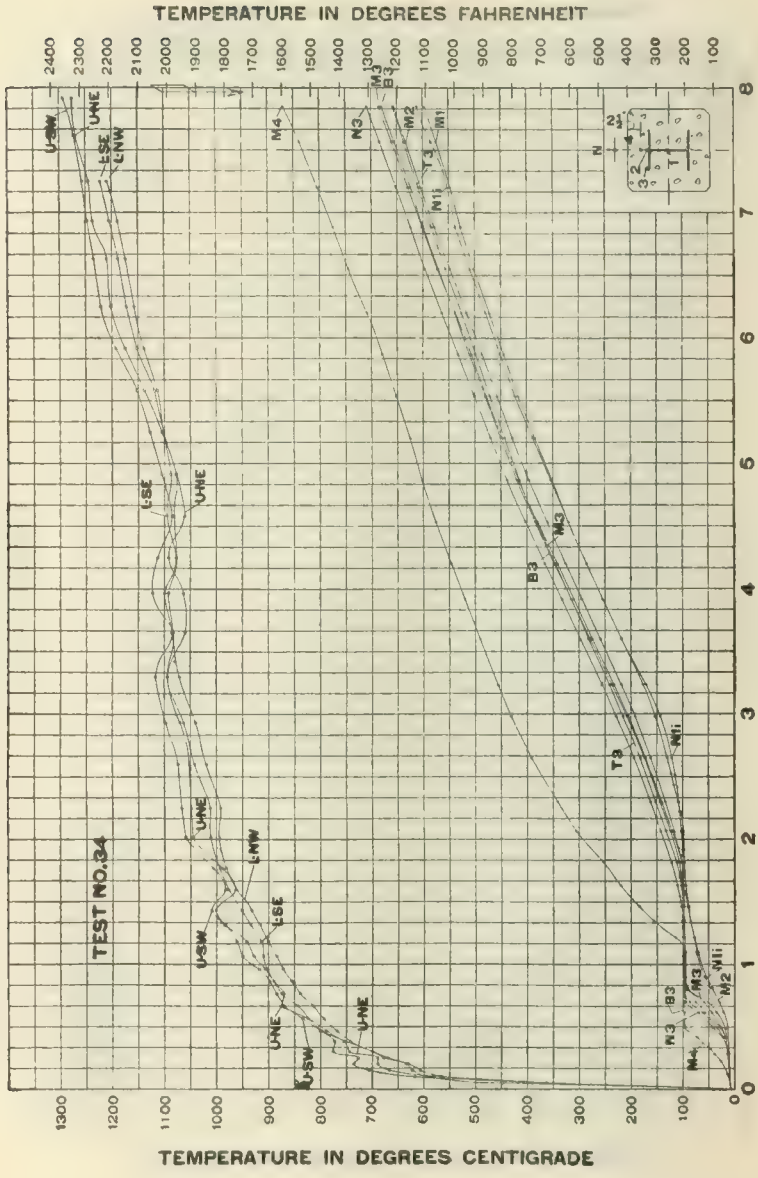


FIG. 107.—*Furnace and column temperatures, 4-inch granite concrete protection, test No. 34*

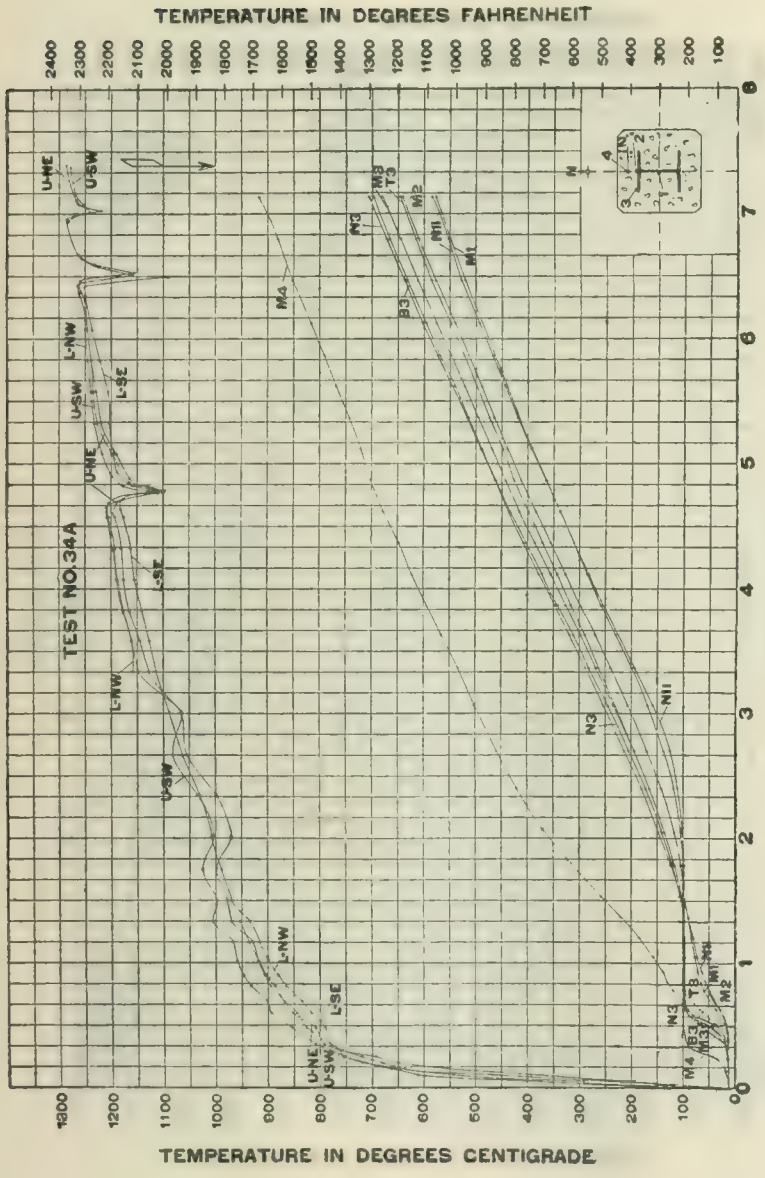


FIG. 108.—*Furnace and column temperatures, 4-inch granite concrete protection, test No. 344*

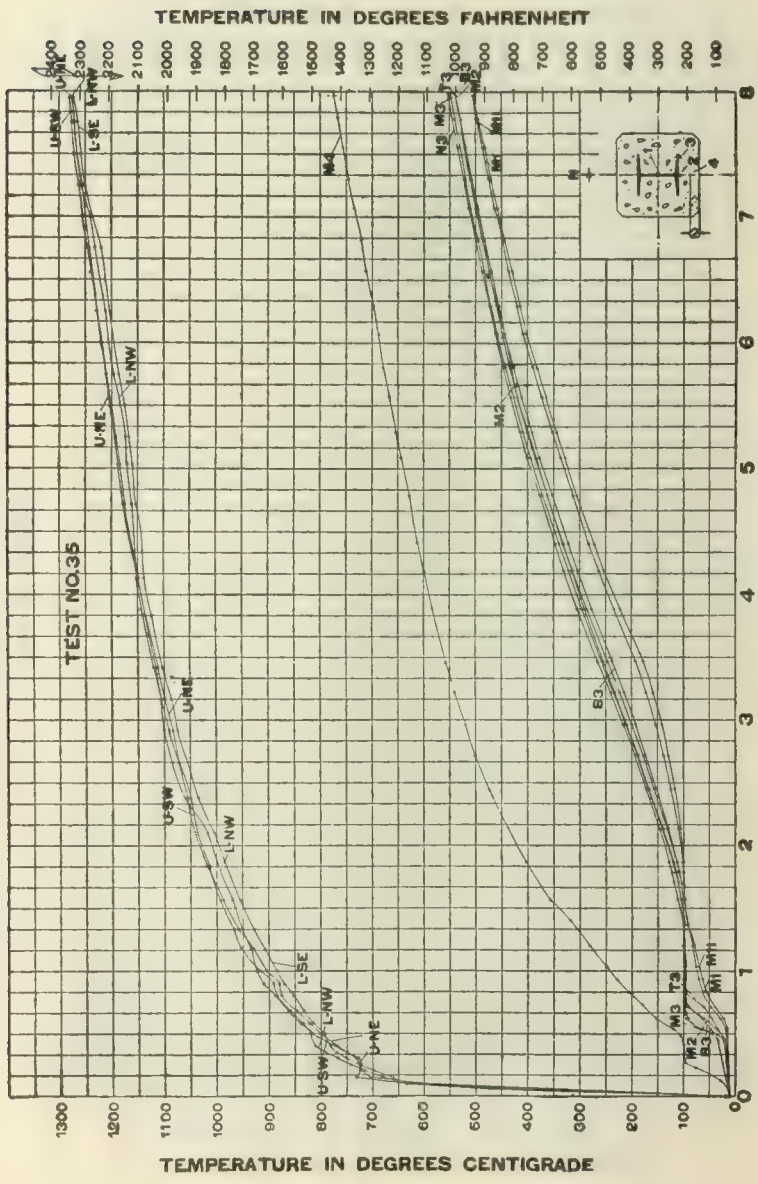


FIG. 109.—*Furnace and column temperatures, 4-inch limestone concrete protection, test No. 35*

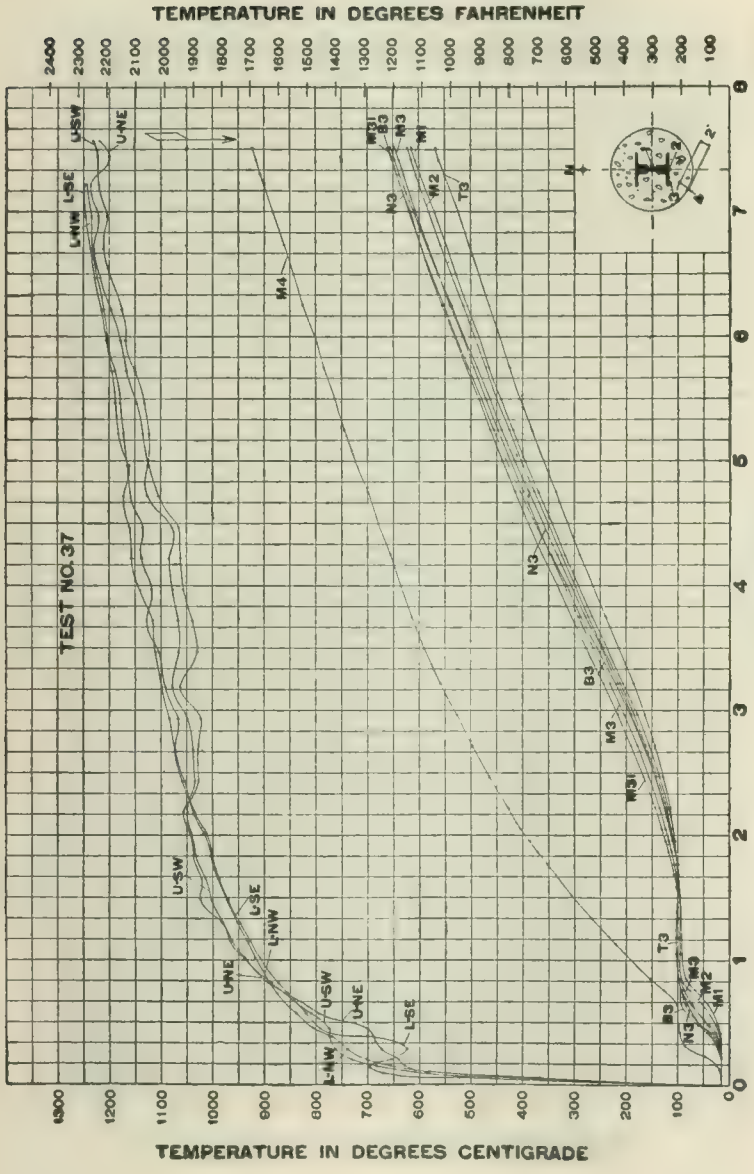


FIG. 110.—*Furnace and column temperatures, 4-inch trap-rock concrete protection, test No. 37*

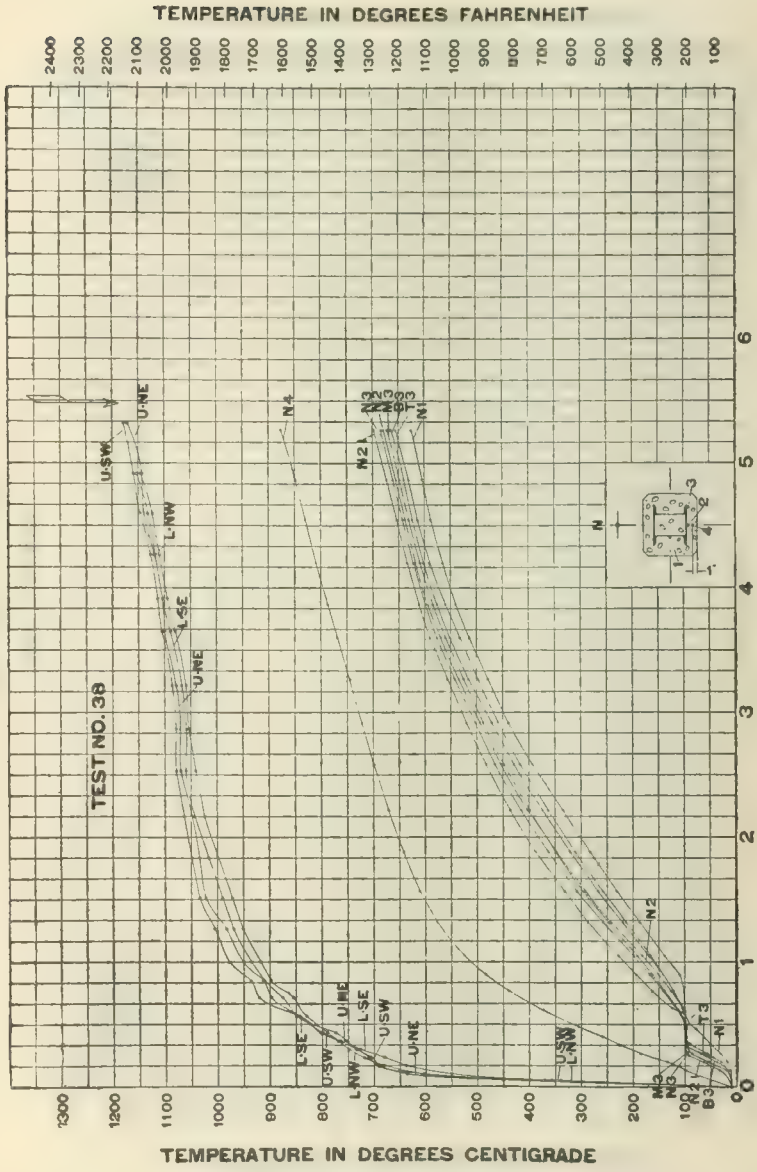


FIG. 111.—*Furnace and column temperatures, 2-inch Joliet gravel concrete protection, test No. 38*

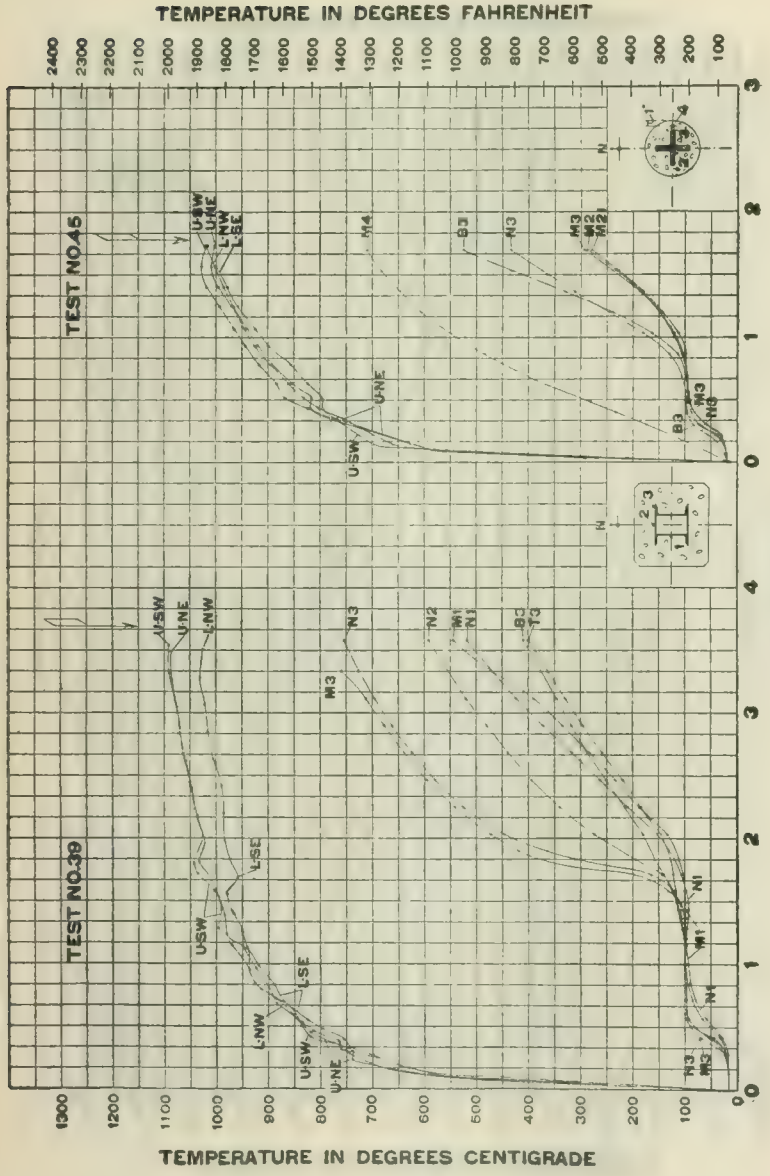


FIG. 112.—*Furnace and column temperatures, 4-inch Meramec River gravel concrete protection, test No. 39; 2-inch Meramec River gravel concrete protection, test No. 45*

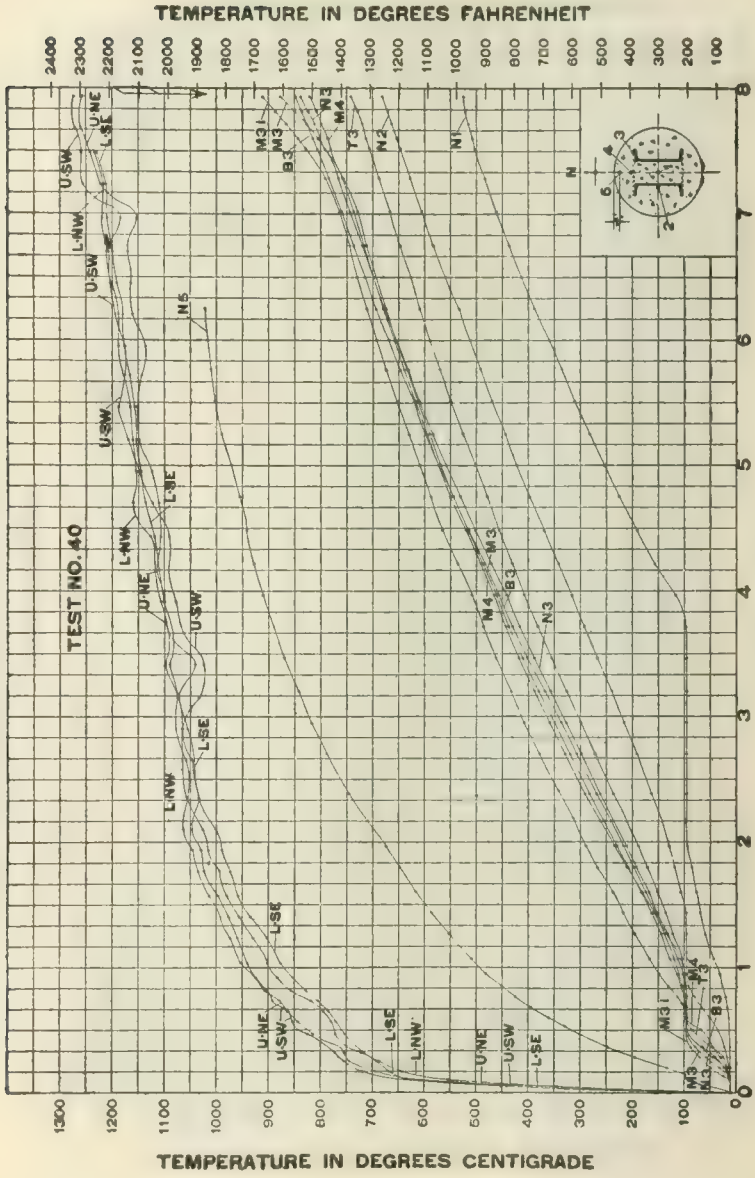


FIG. 113.—*Furnace and column temperatures, 2-inch trap-rock concrete protection, test No. 40*

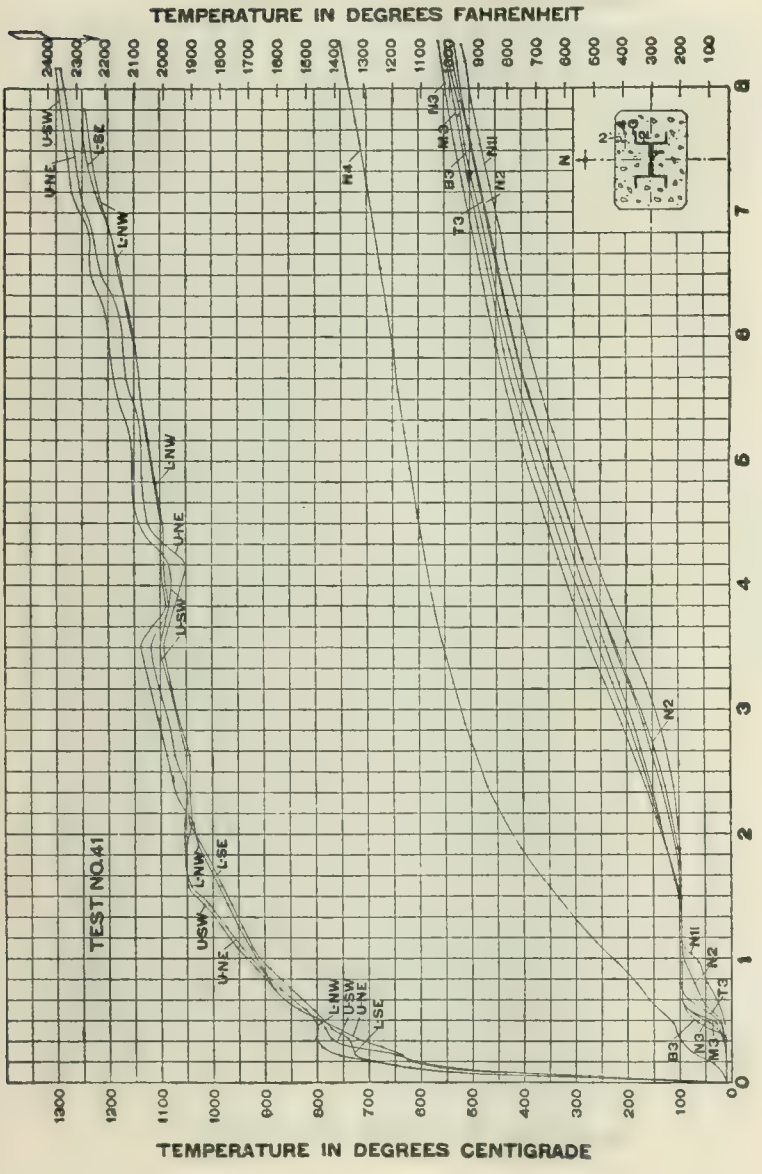


FIG. 114.—*Furnace and column temperatures, 4-inch limestone concrete protection, test No. 41*

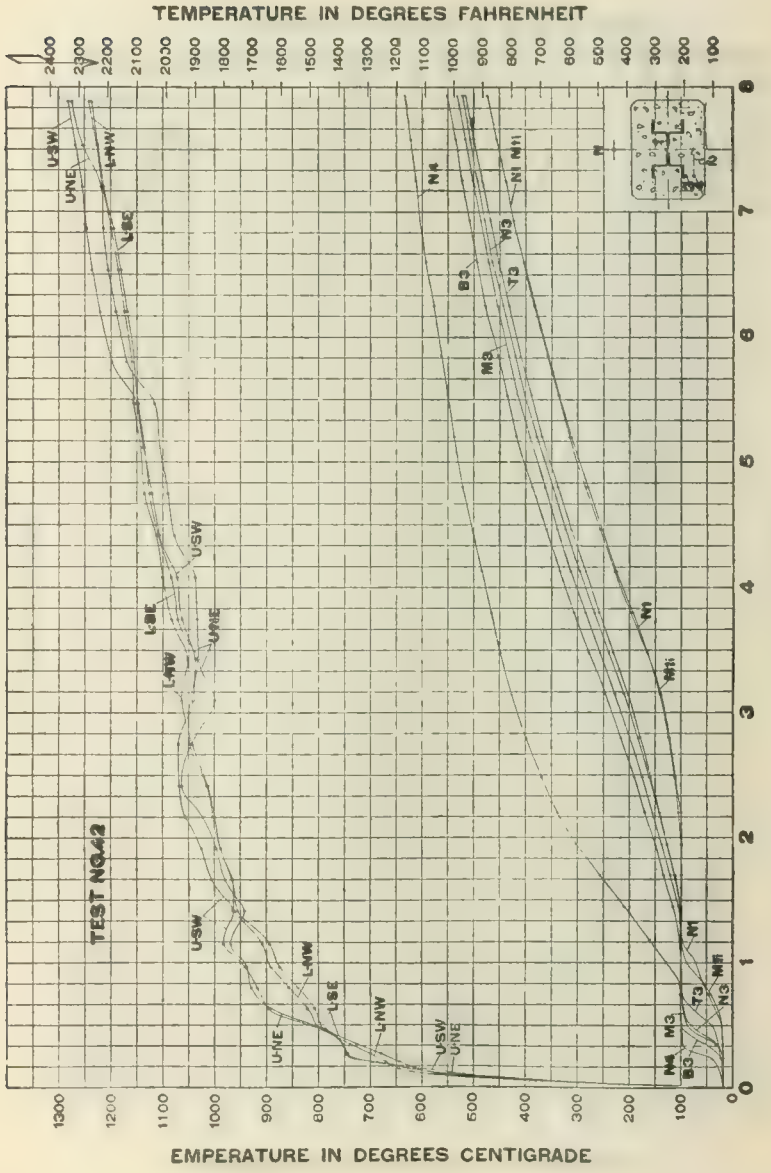


FIG. 115.—*Furnace and column temperatures, 4-inch limestone concrete protection, test No. 42*

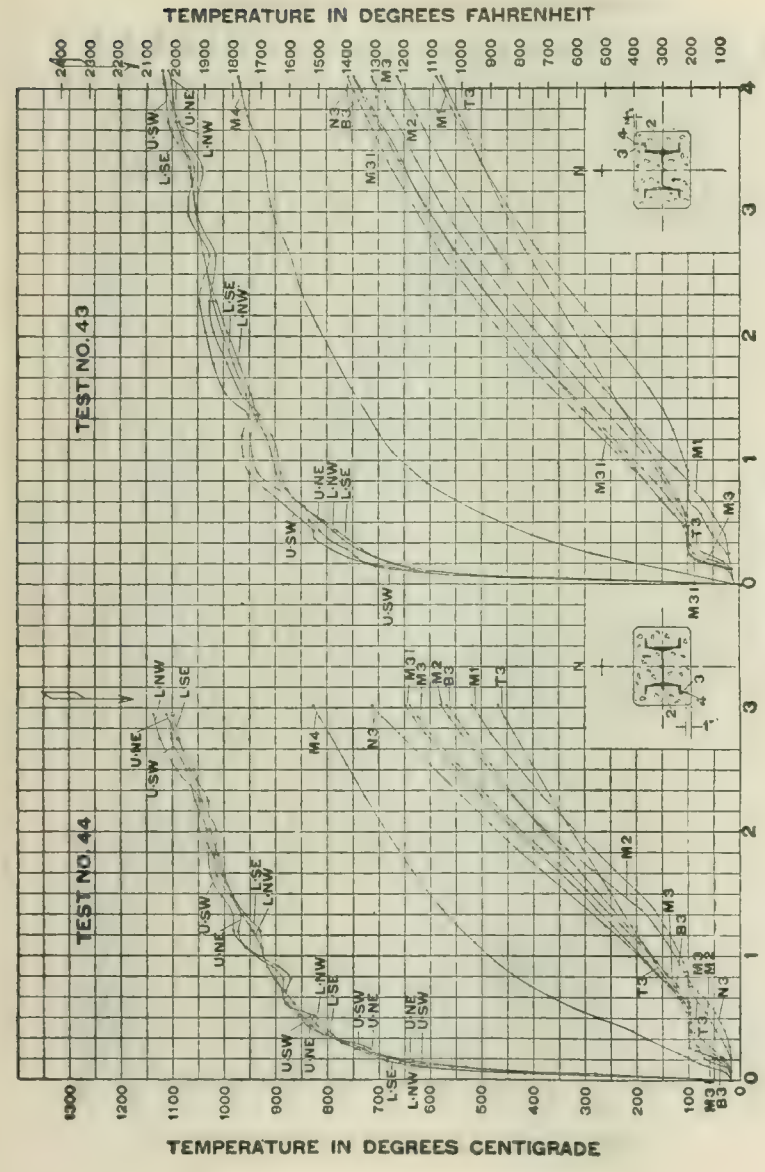


FIG. 116.—*Furnace and column temperatures, 2-inch sandstone concrete protections, test Nos. 43 and 44*

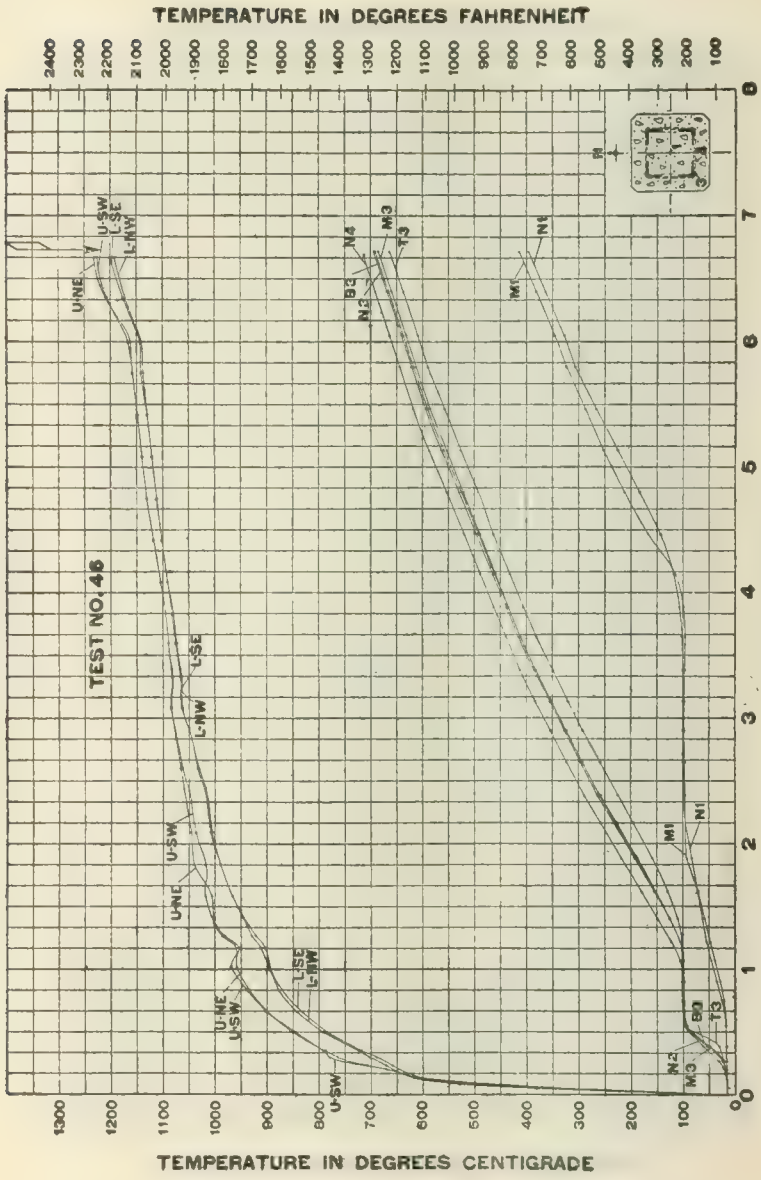


FIG. 117.—*Furnace and column temperatures, 2-inch trap-rock concrete protection (3⅛-inch of outside angles), test No. 46*

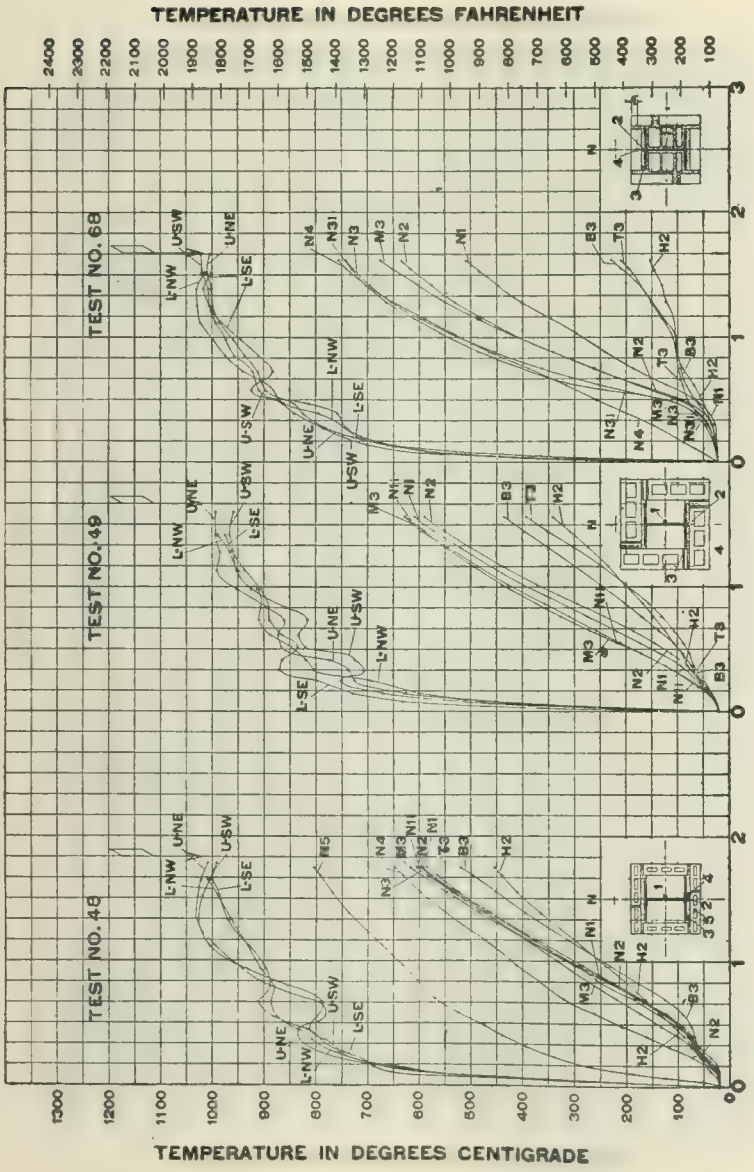


FIG. 118.—*Furnace and column temperatures, semifire-clay tile protections, test Nos. 48 and 49; 2¼-inch surface clay brick protection, test No. 68*

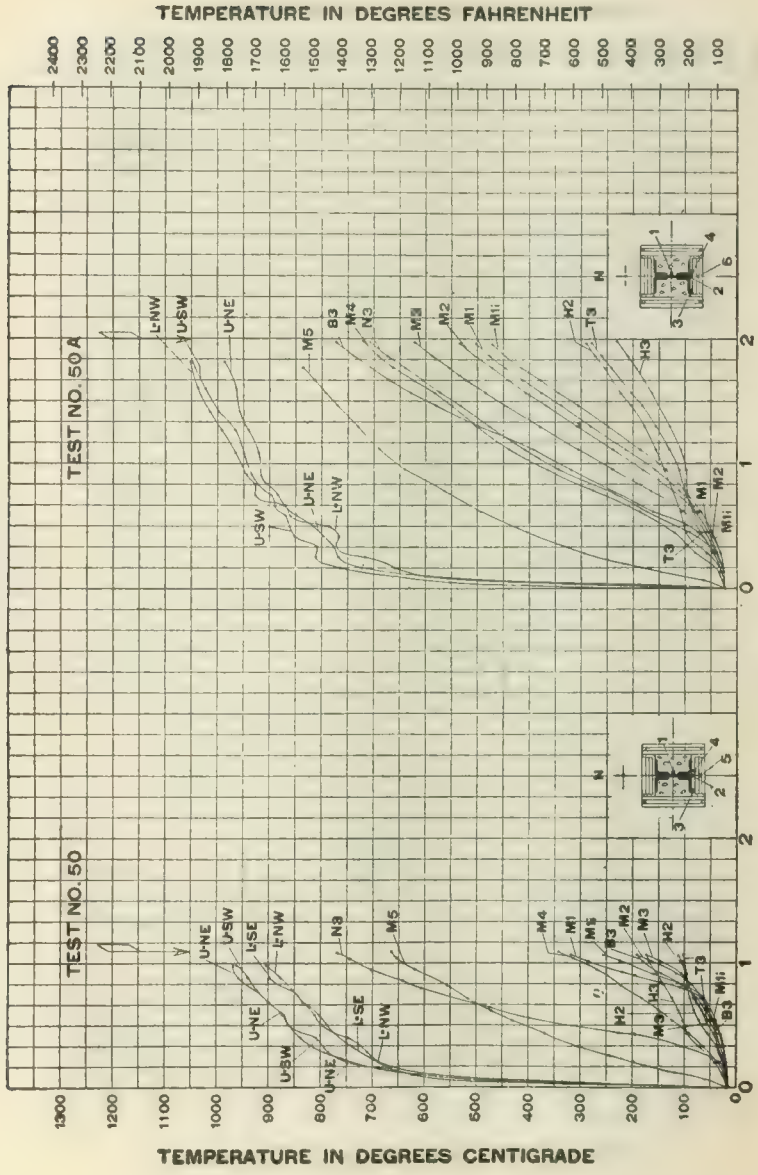


FIG. 119.—*Furnace and column temperatures, surface-clay tile protections, test Nos. 50 and 50A*

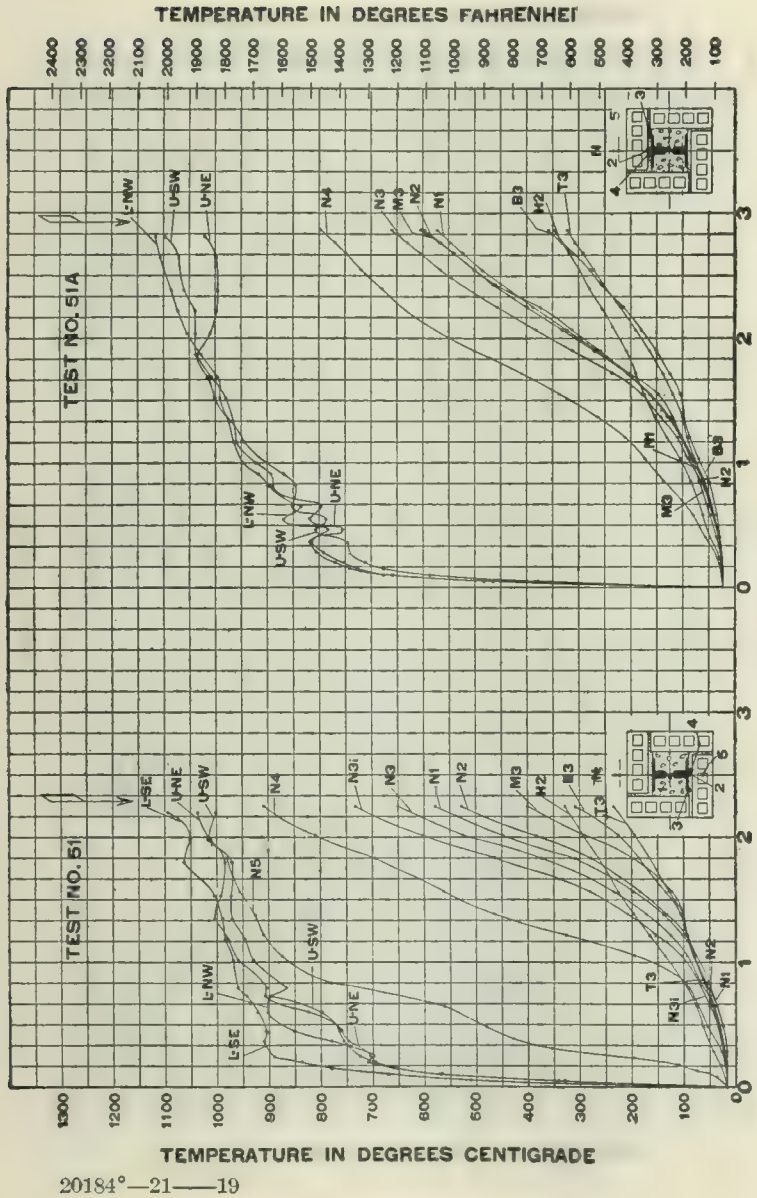


FIG. 120.—*Furnace and column temperatures, surface-clay tile protections, test Nos. 51 and 51A*

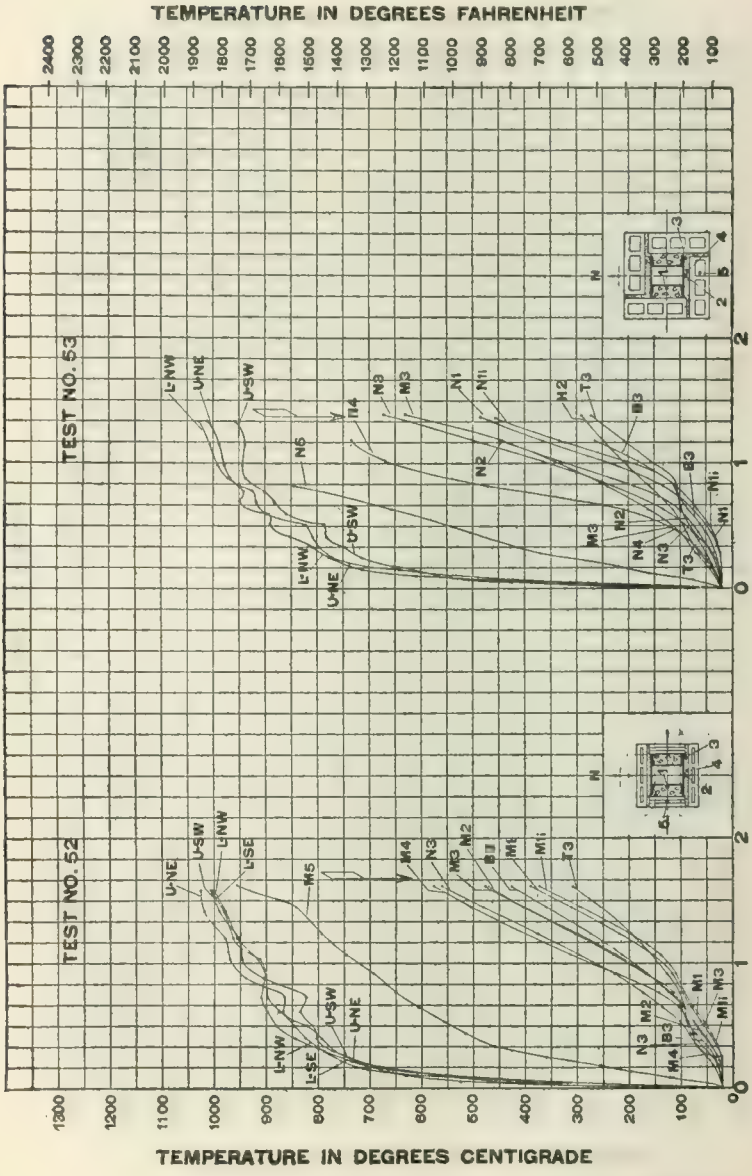


FIG. 121.—*Furnace and column temperatures, shale tile protections, test Nos. 52 and 53*

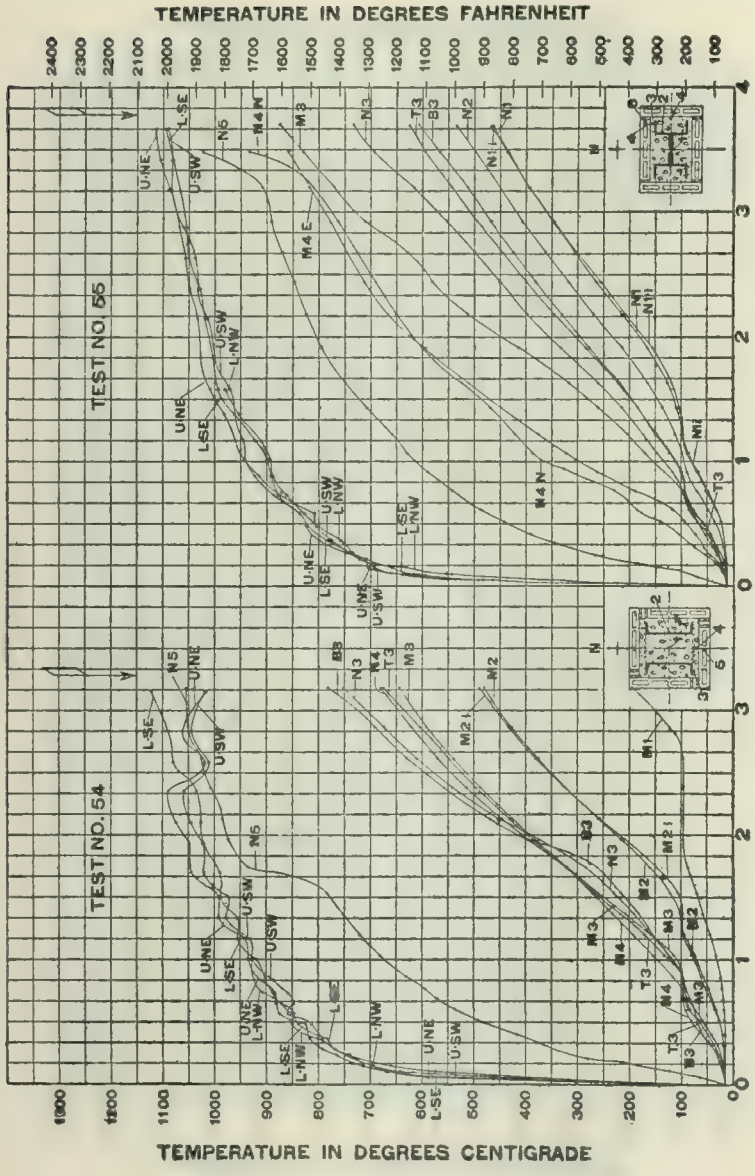


FIG. 122.—*Furnace and column temperatures, semifire-clay tile protections, test Nos. 54 and 55*

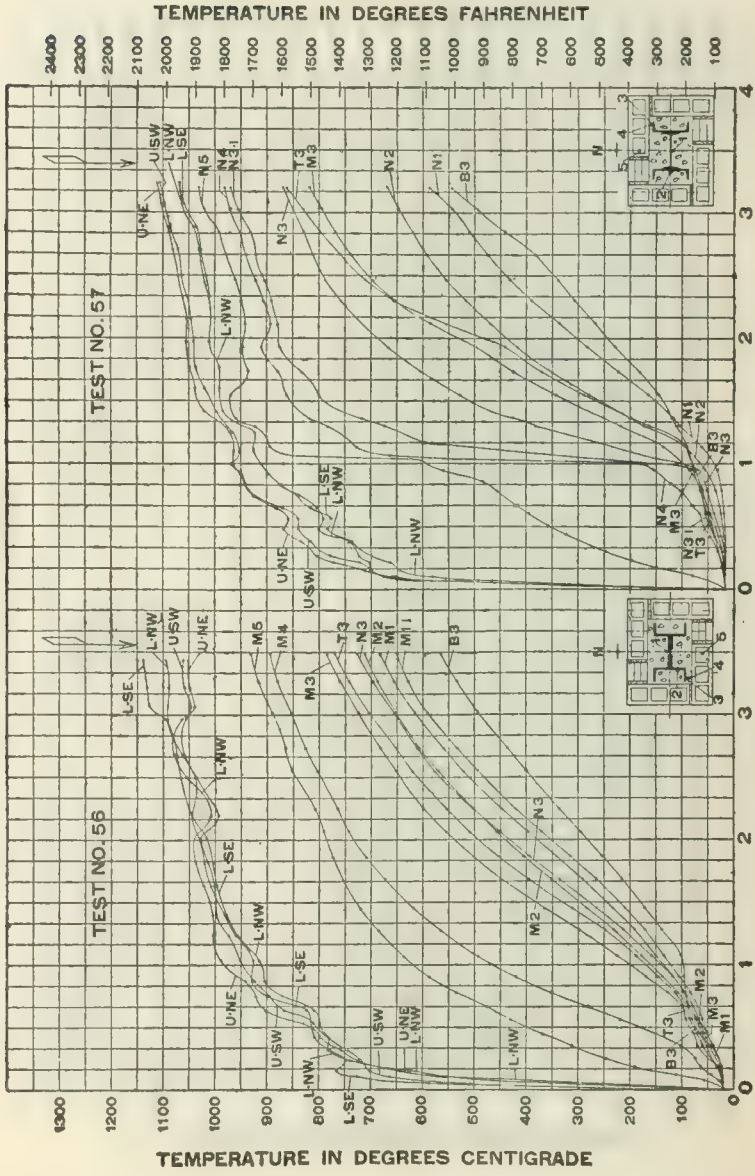


FIG. 123.—*Furnace and column temperatures, semifire-clay tile protection, test No. 56; surface-clay tile protection, test No. 57*

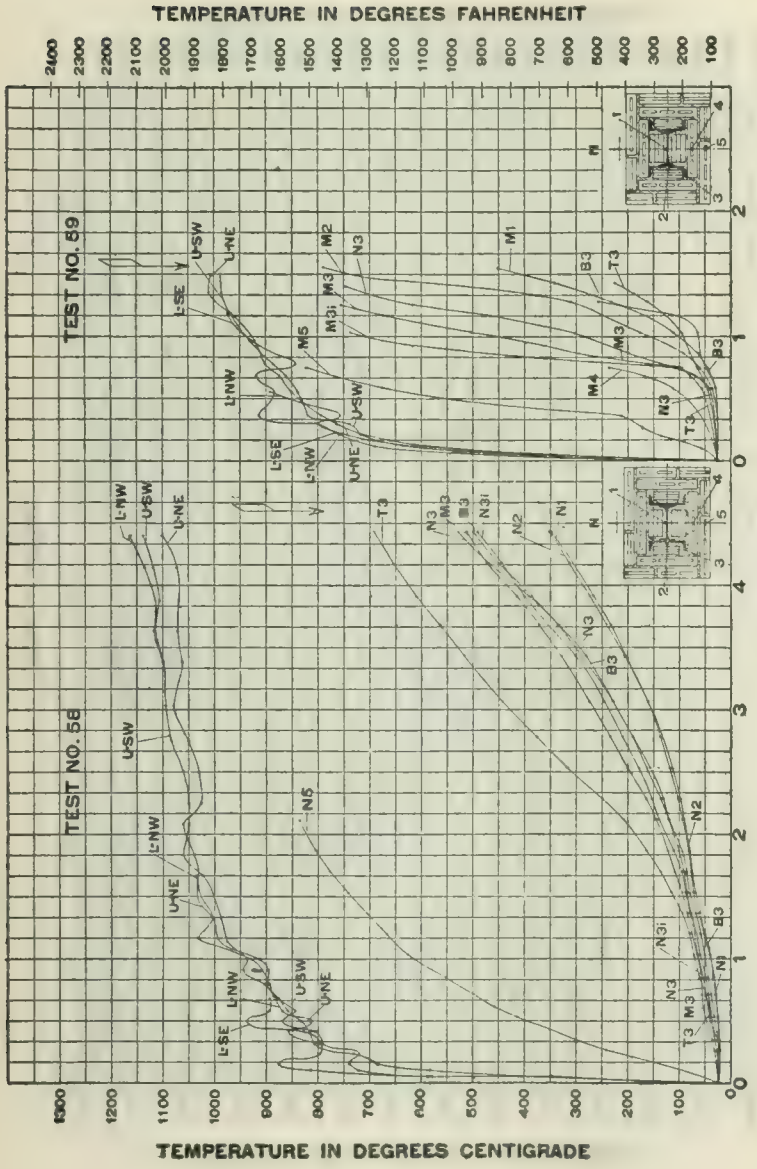


FIG. 124.—*Furnace and column temperatures, surface-clay tile protections, test Nos. 58 and 59*

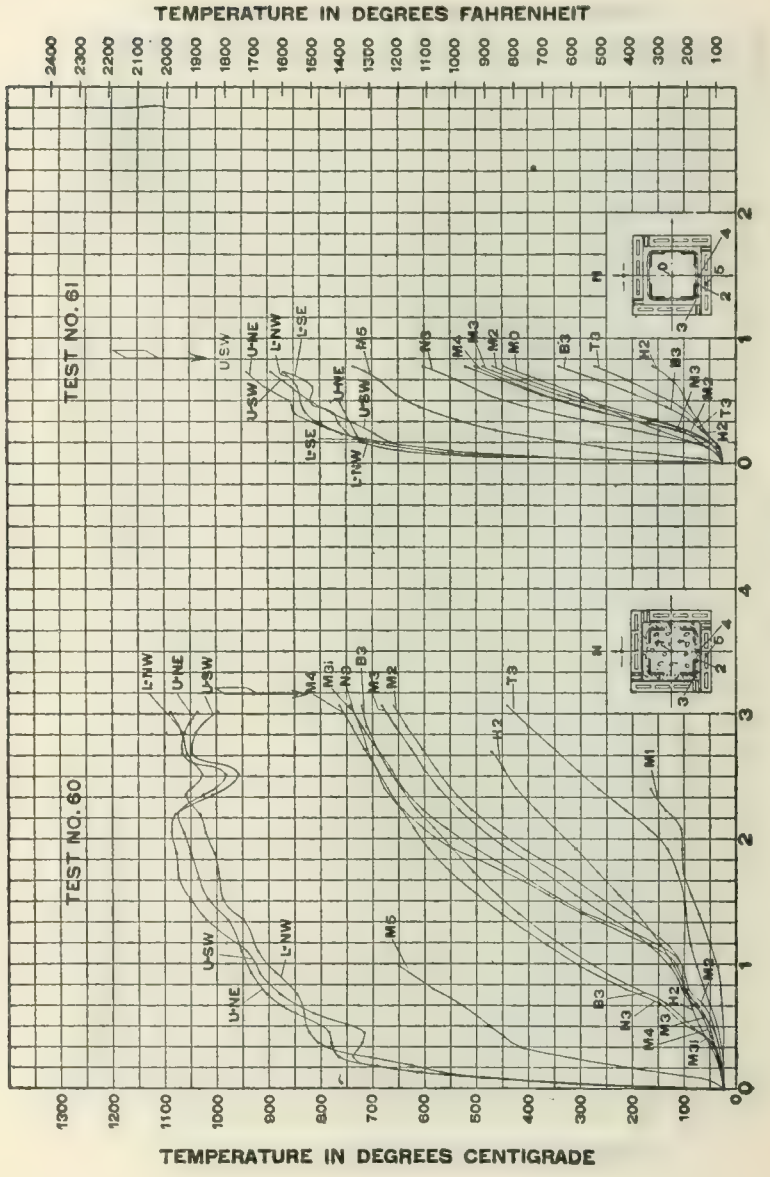


FIG. 125.—*Furnace and column temperatures, semifire-clay tile protections, test Nos. 60 and 61*

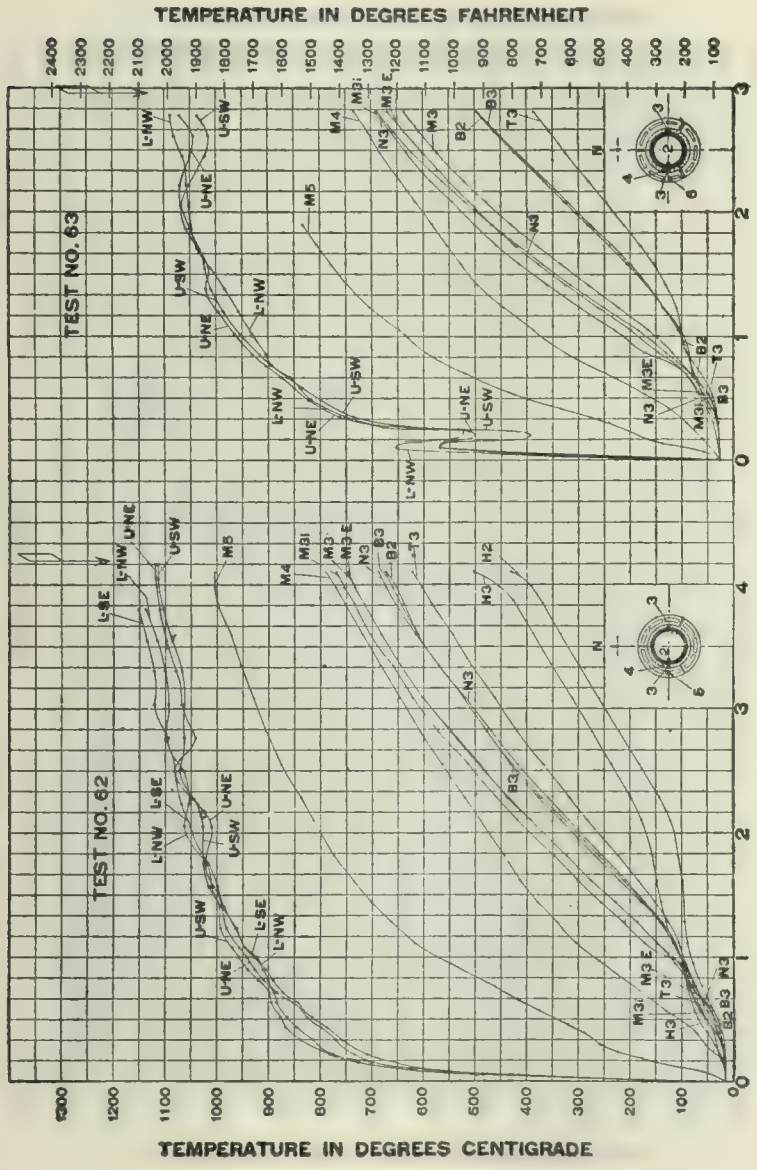


FIG. 126.—*Furnace and column temperatures, porous semifire-clay tile protections, test Nos. 62 and 63*

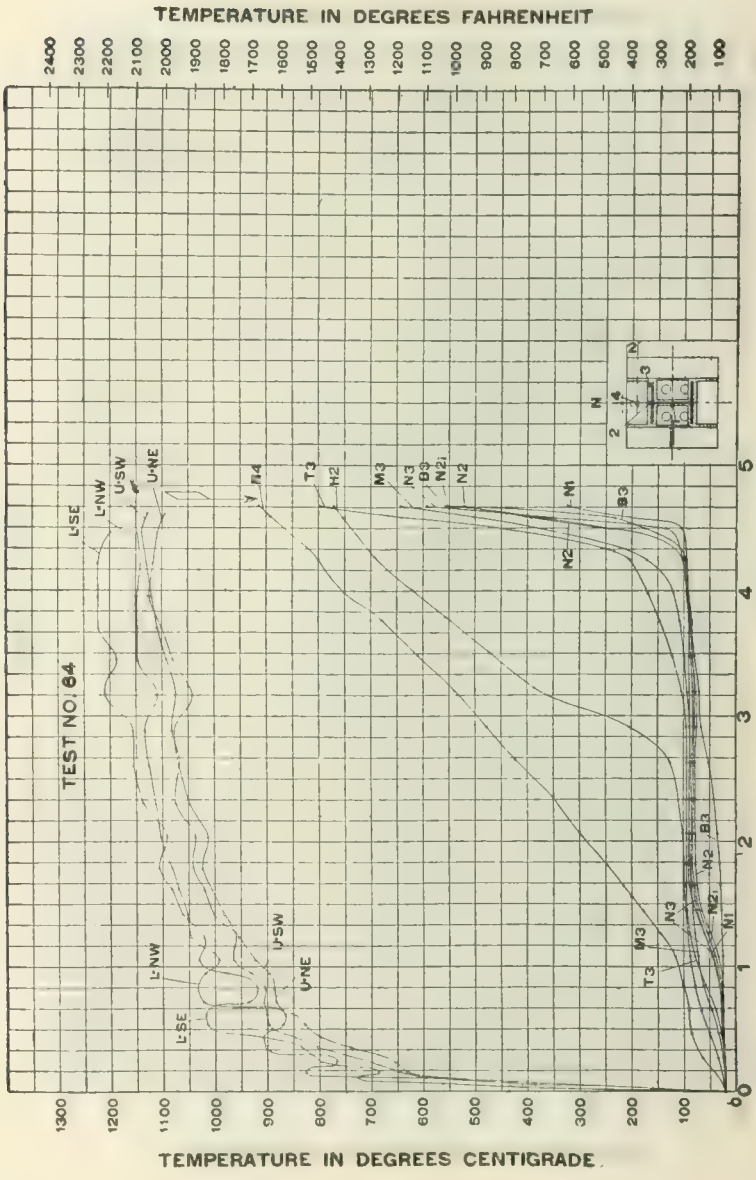


FIG. 127.—*Furnace and column temperatures, 4-inch gypsum-block protection, test No. 64*

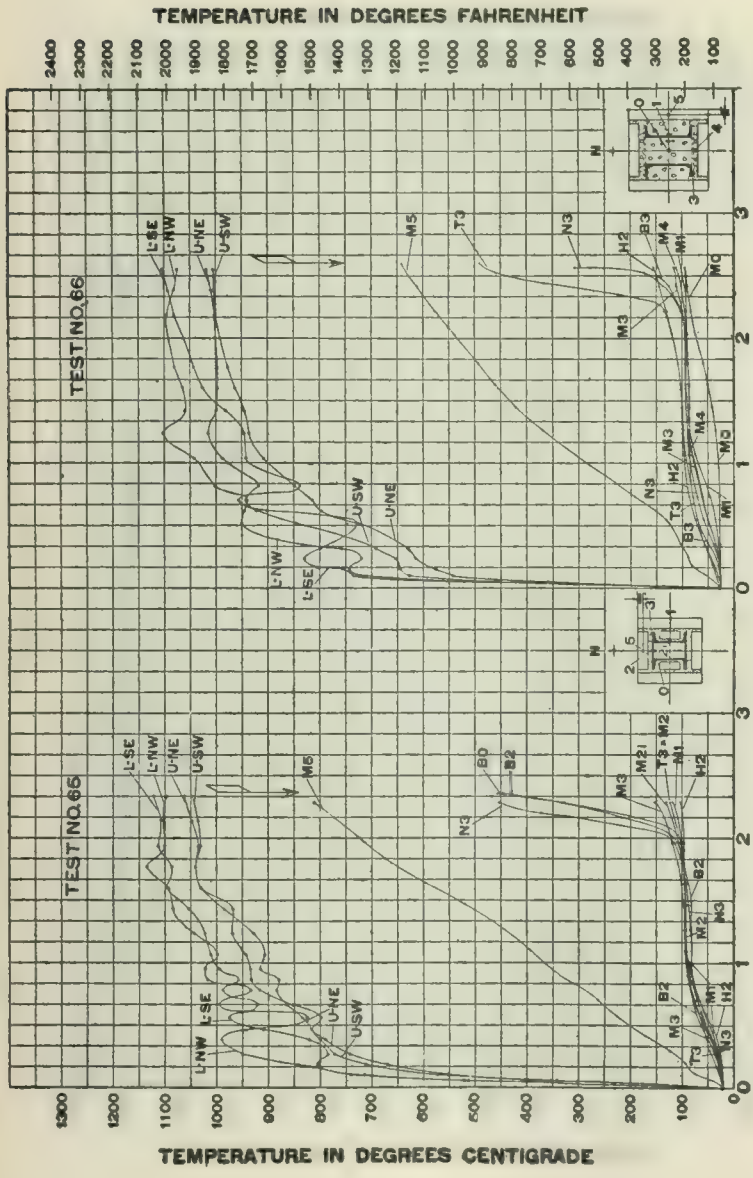


FIG. 128.—*Furnace and column temperatures, 2-inch gypsum-block protections, test Nos. 65 and 66*

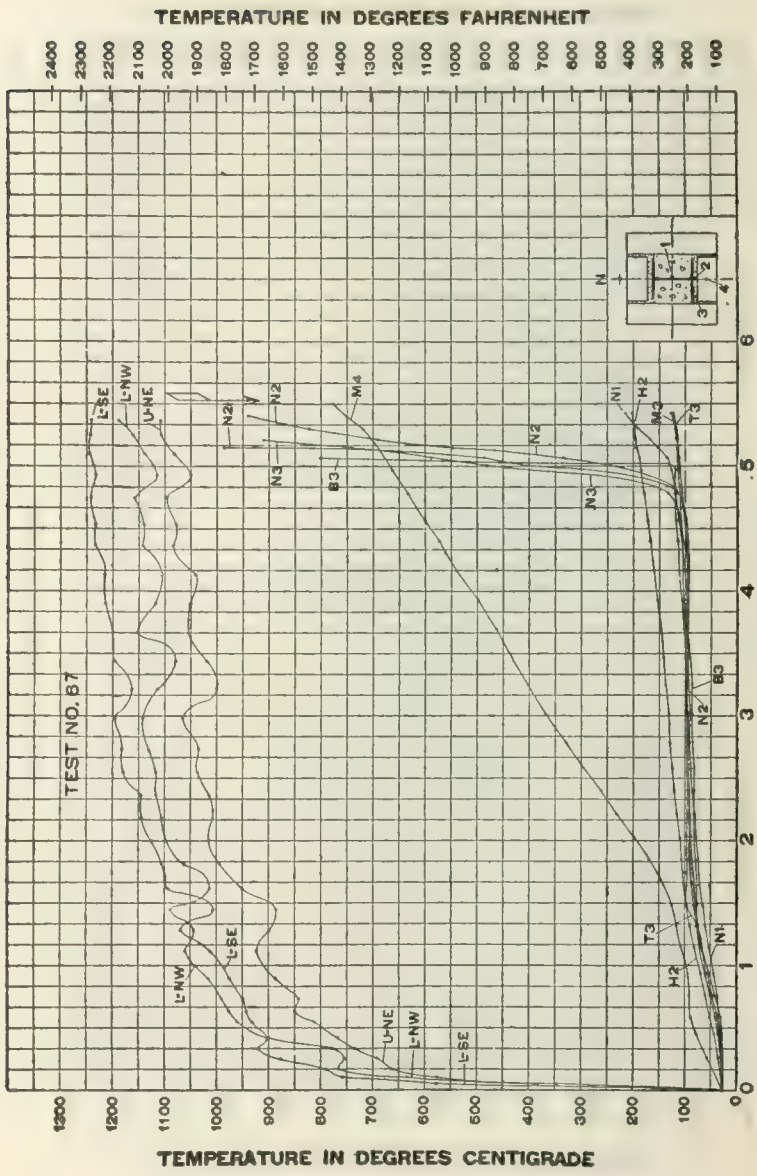


FIG. 129.—*Furnace and column temperatures, 4-inch gypsum-block protection, test No. 67*

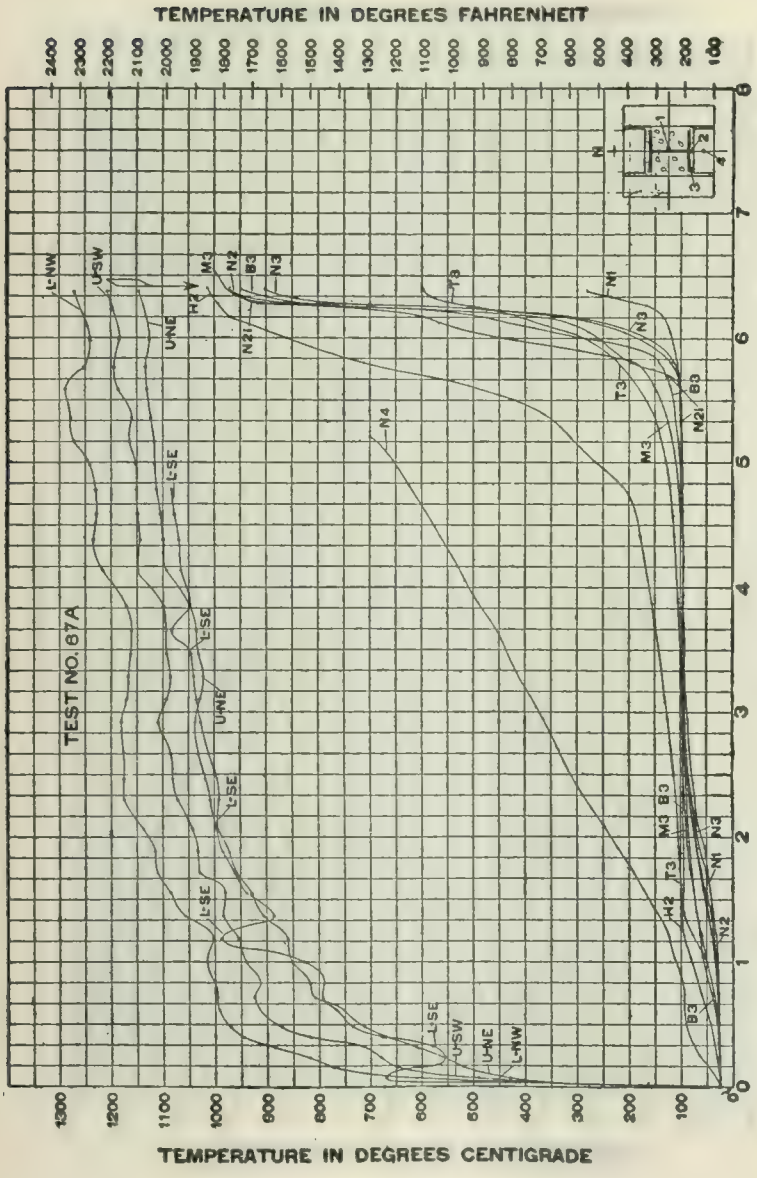


FIG. 130.—*Furnace and column temperatures, 4-inch gypsum-block protection, test No. 67A*

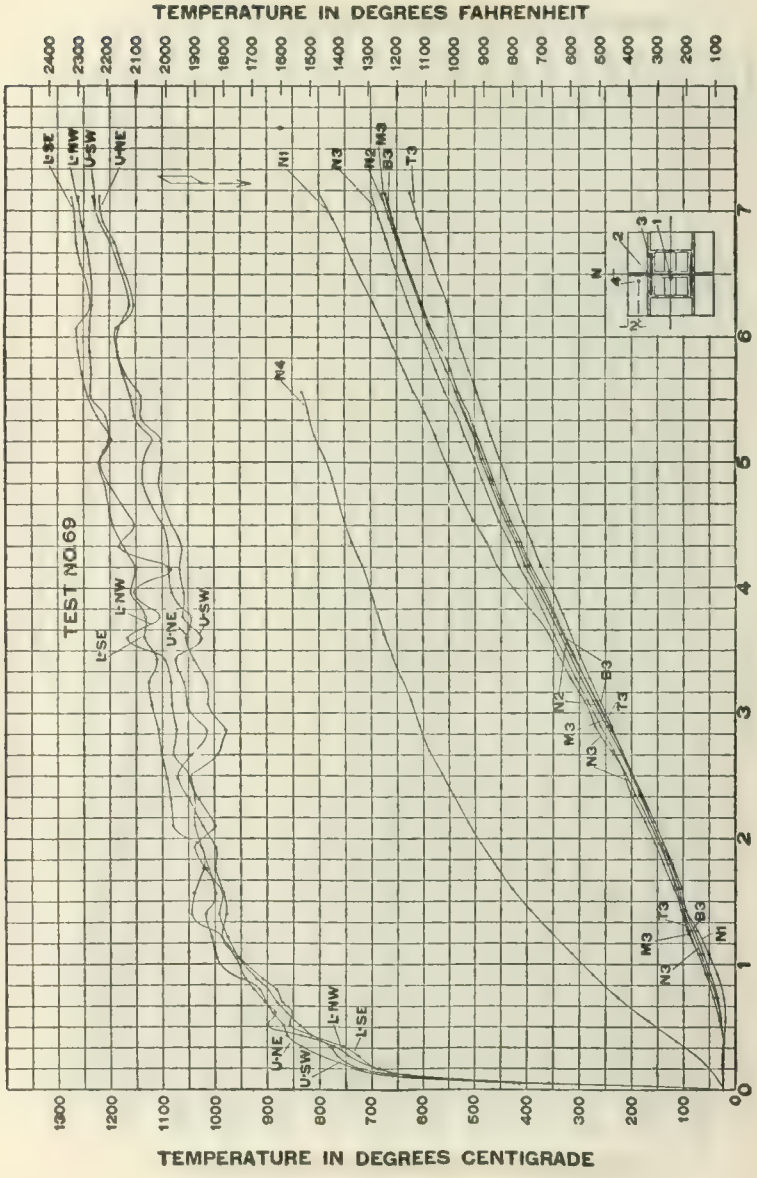


FIG. 131.—*Furnace and column temperatures, 3¼-inch surface-clay brick protection, test No. 69*

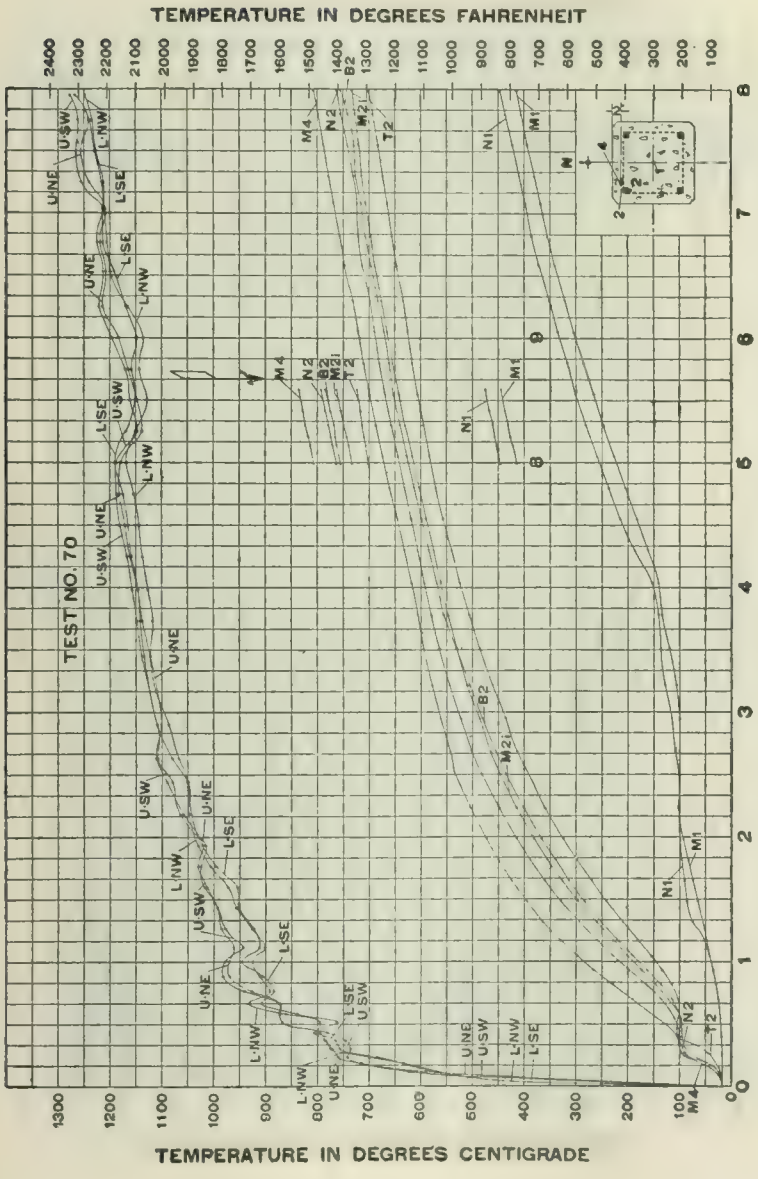


FIG. 132.—*Furnace and column temperatures, square, reinforced, limestone concrete column, test No. 70*

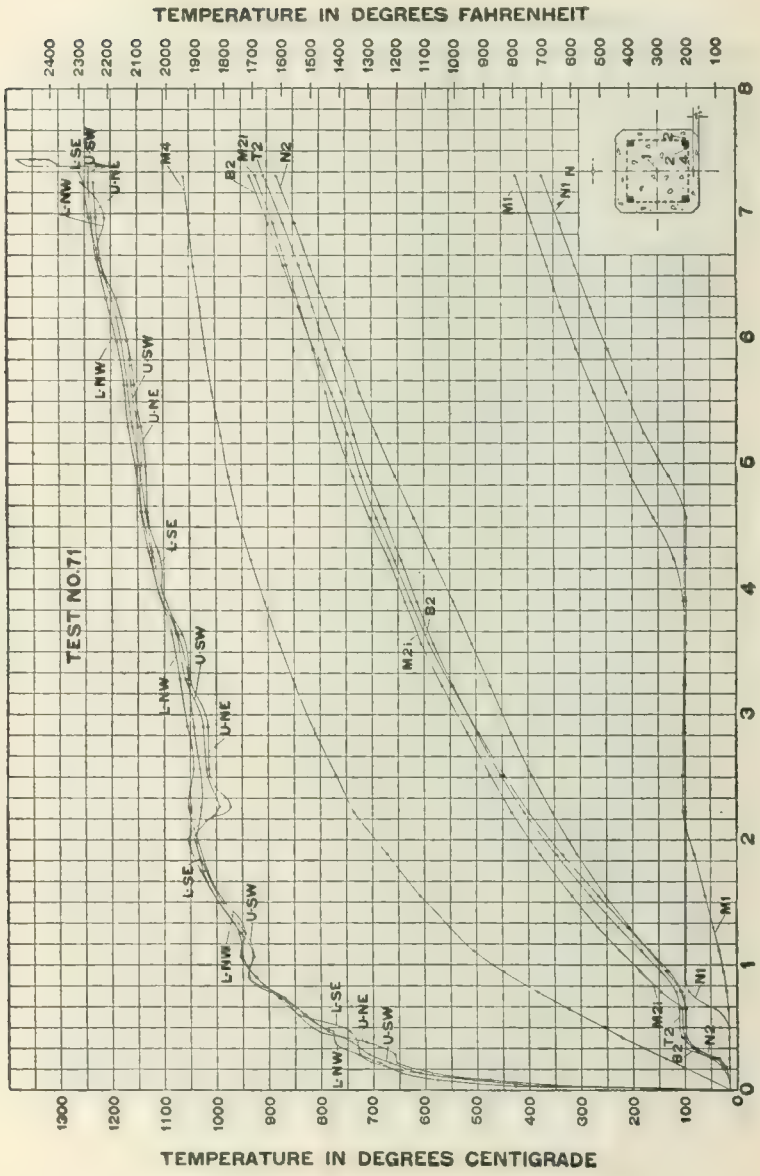


FIG. 133.—*Furnace and column temperatures, square, reinforced, trap-rock concrete column, test No. 71*

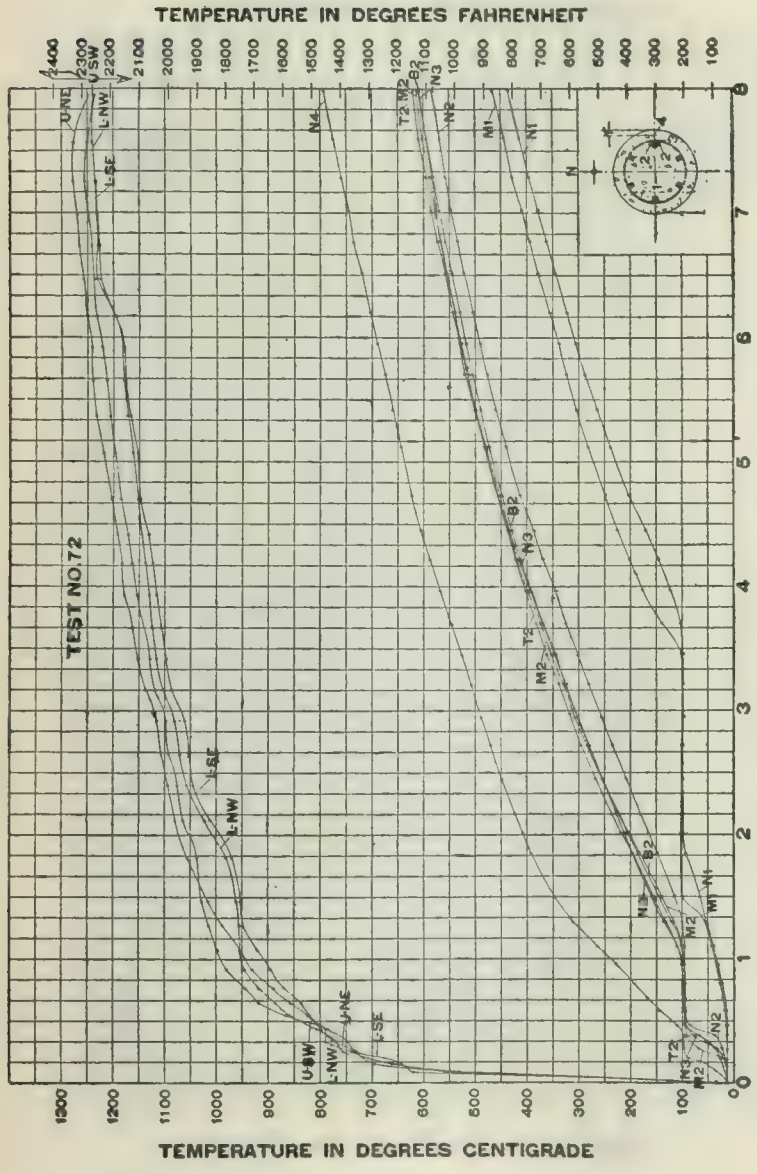


FIG. 134.—*Furnace and column temperatures, round, reinforced, limestone concrete columns, test No. 72*

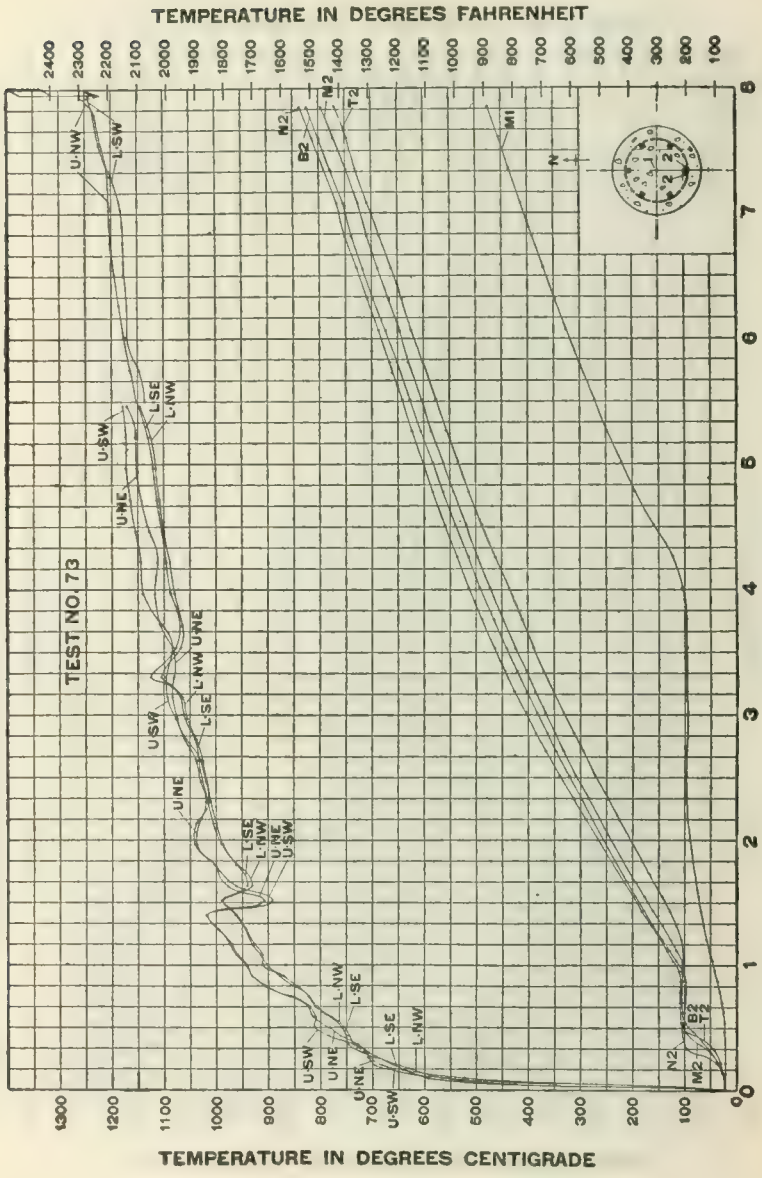


FIG. 135.—*Furnace and column temperatures, round, reinforced, trap-rock concrete column, test No. 73*

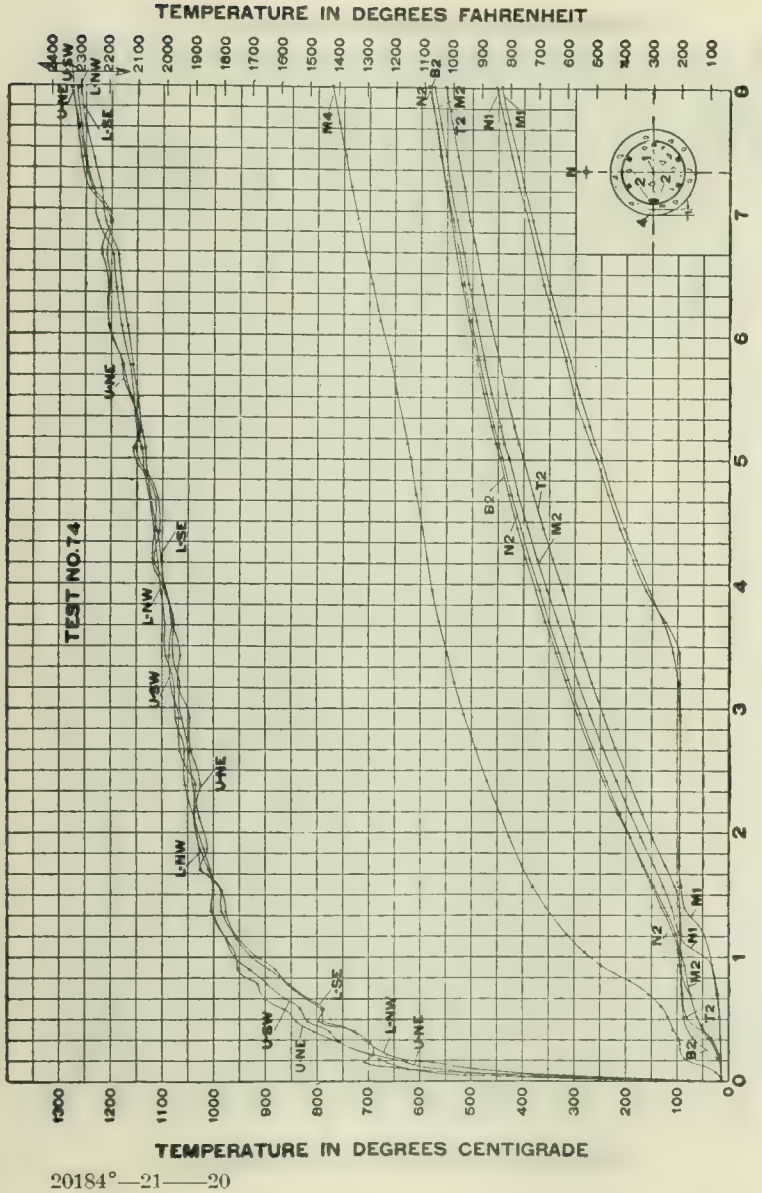


FIG. 136.—*Furnace and column temperatures, round, hooped, limestone concrete column, test No. 74*

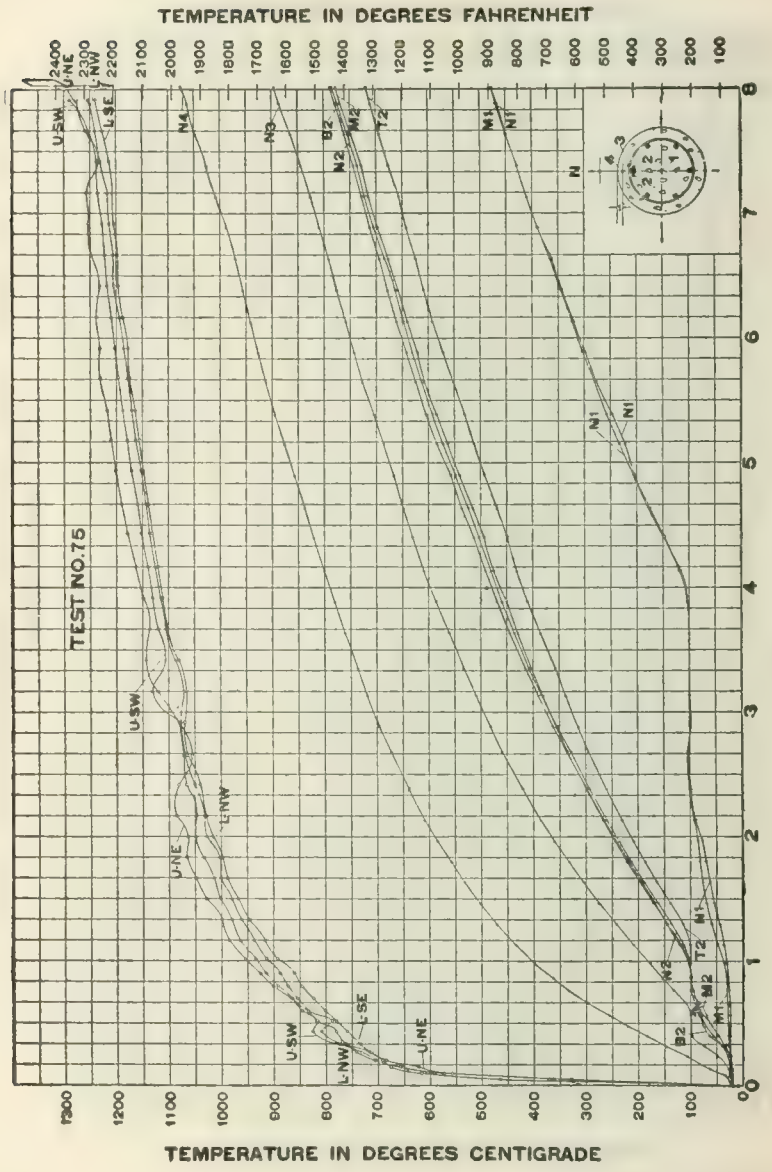


FIG. 137.—*Furnace and column temperatures, round, hooped, trap-rock concrete column, test No. 75*

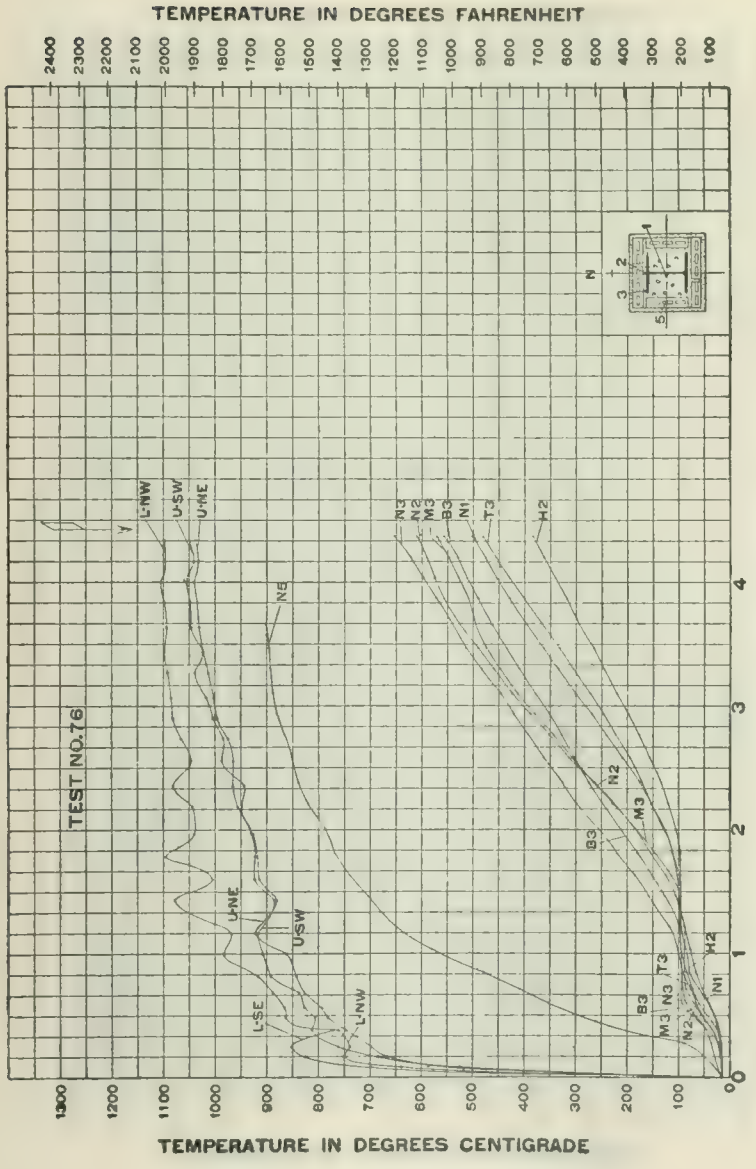


FIG. 138.—*Furnace and column temperatures, plastered clay-tile protection, test No. 76*

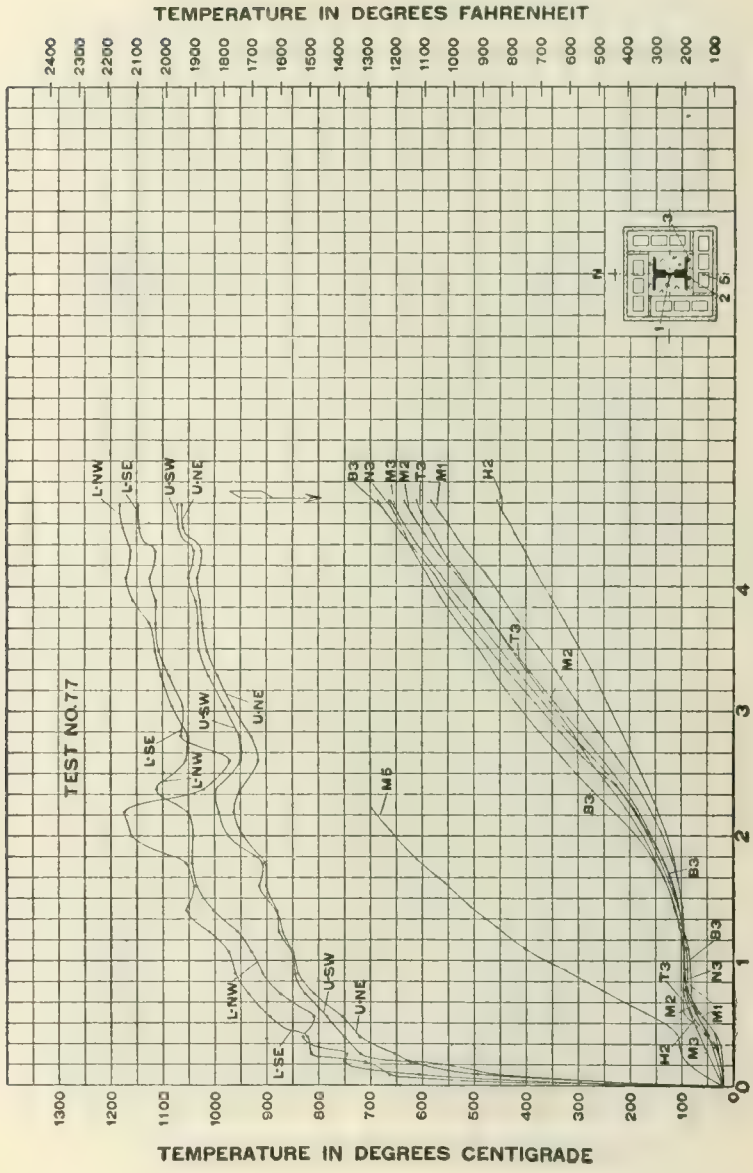


FIG. 139.—*Furnace and column temperatures, plastered clay-tile protection, test No. 77*

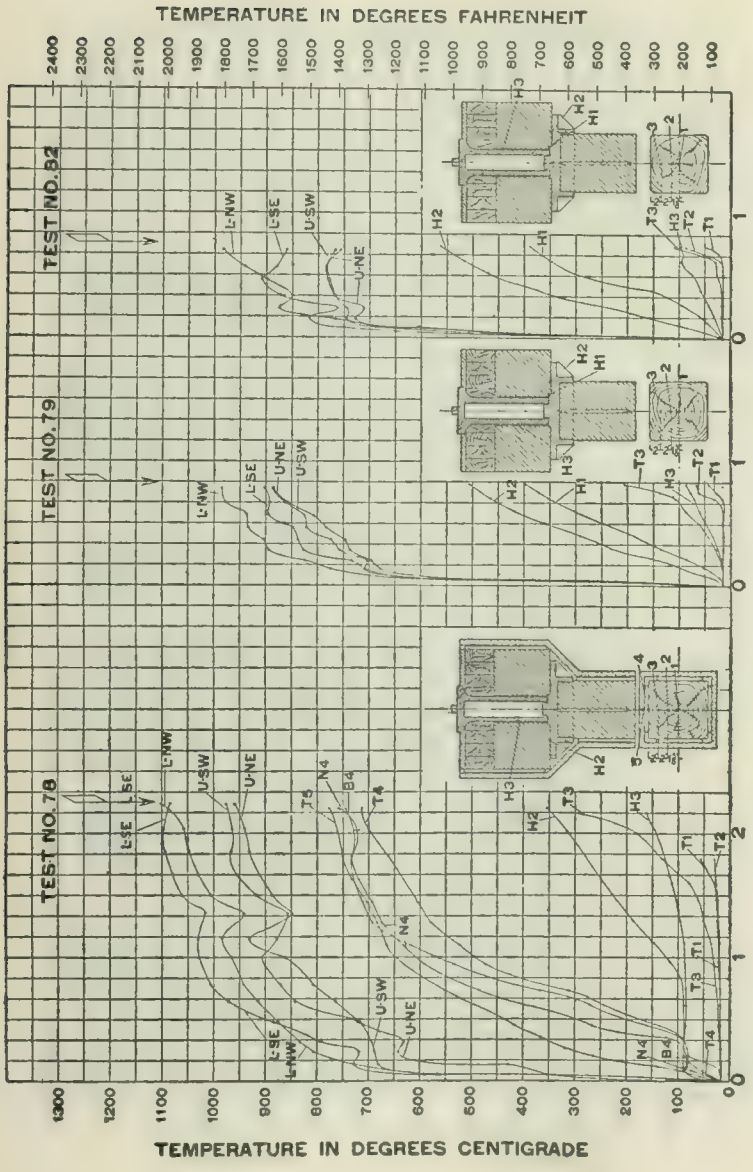


FIG. 140.—*Furnace and column temperatures, timber columns, cap and pintle bearings, test Nos. 78, 79, and 82*

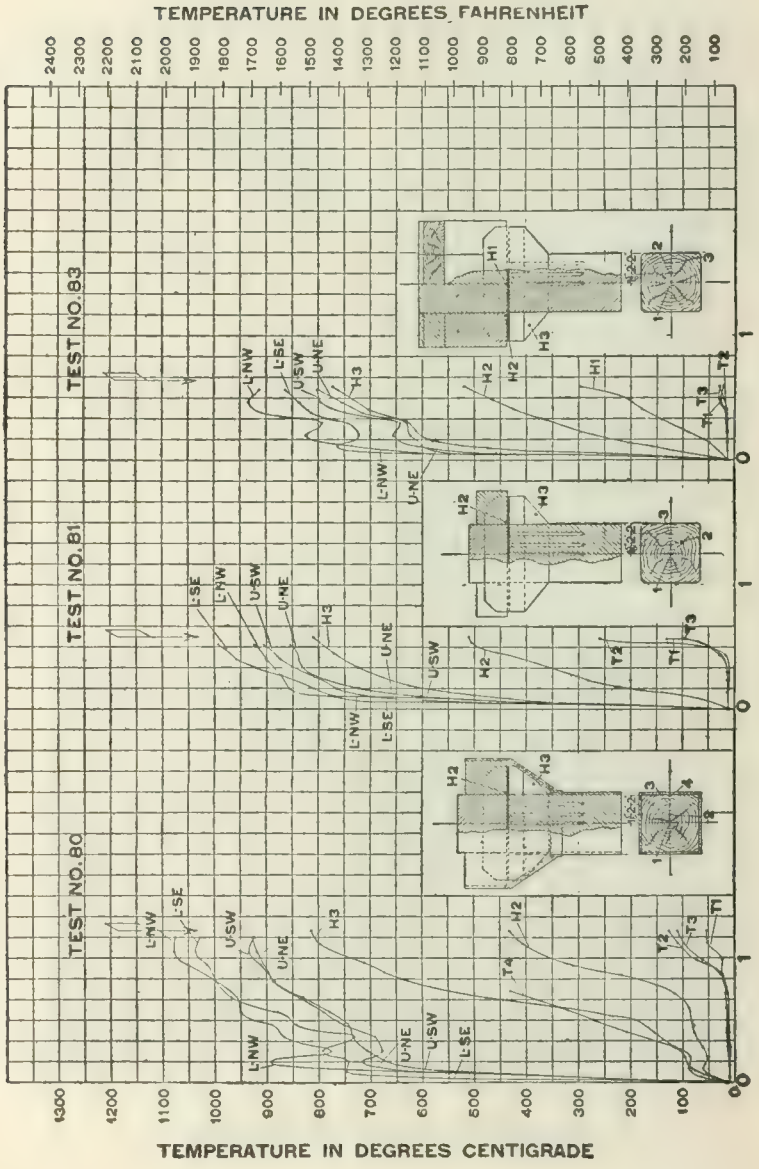


FIG. 141.—*Furnace and column temperatures, timber columns, steel-plate cap bearings, test Nos. 80, 81, and 83*

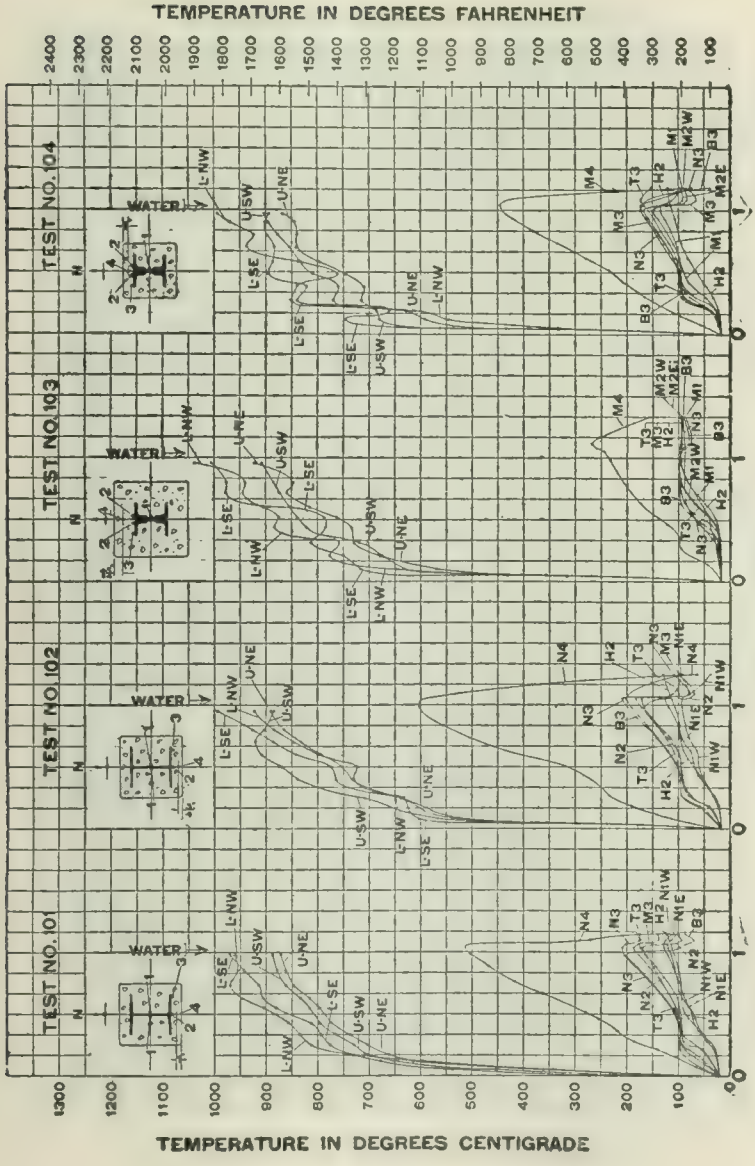


FIG. 142.—*Furnace and column temperatures, concrete protections, fire and water tests, Nos. 101 to 104*

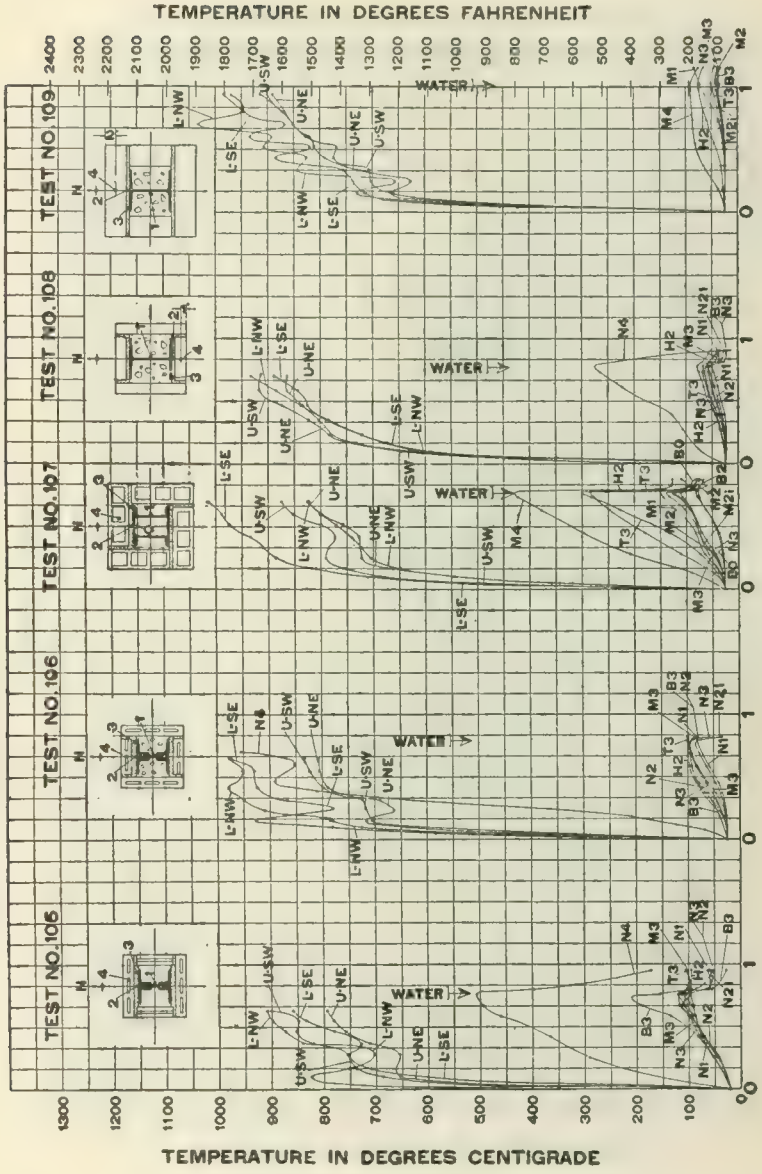


FIG. 143.—*Furnace and column temperatures, clay-tile and gypsum-block protections, fire and water tests, Nos. 105 to 109*

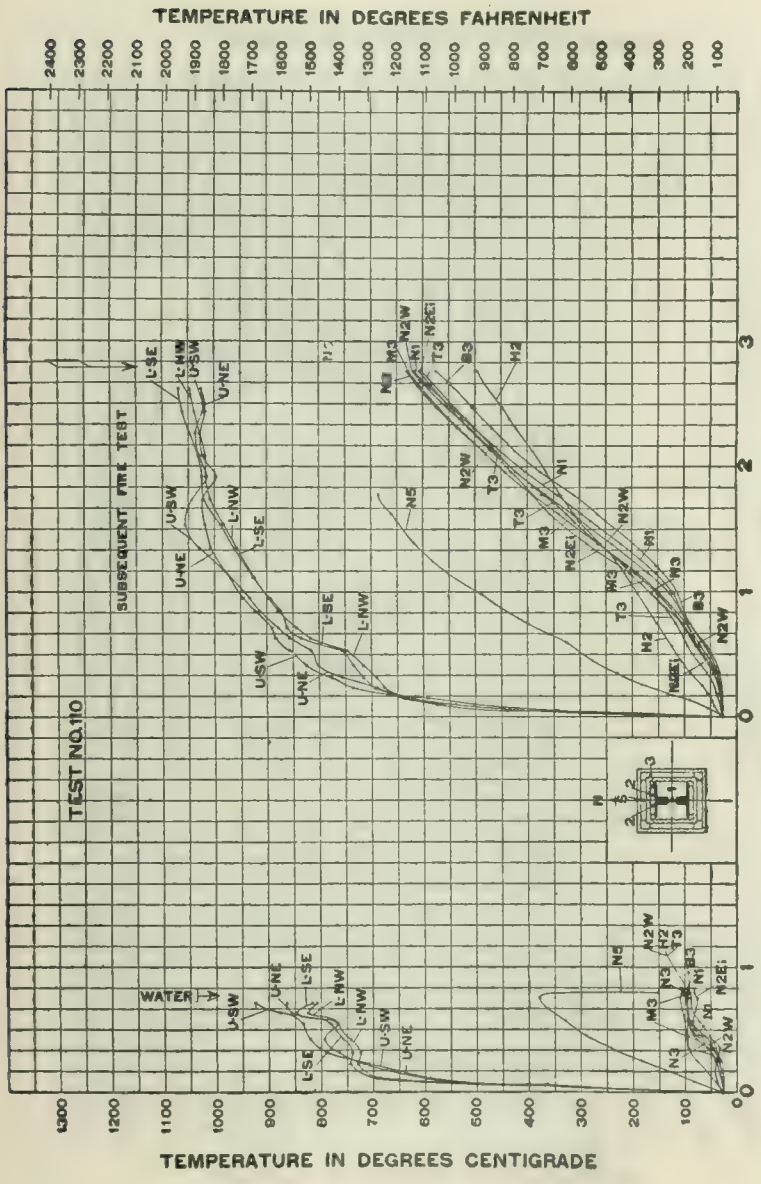


FIG. 144.—*Furnace and column temperatures, plaster on metal-lath protection, fire and water test, No. 110*

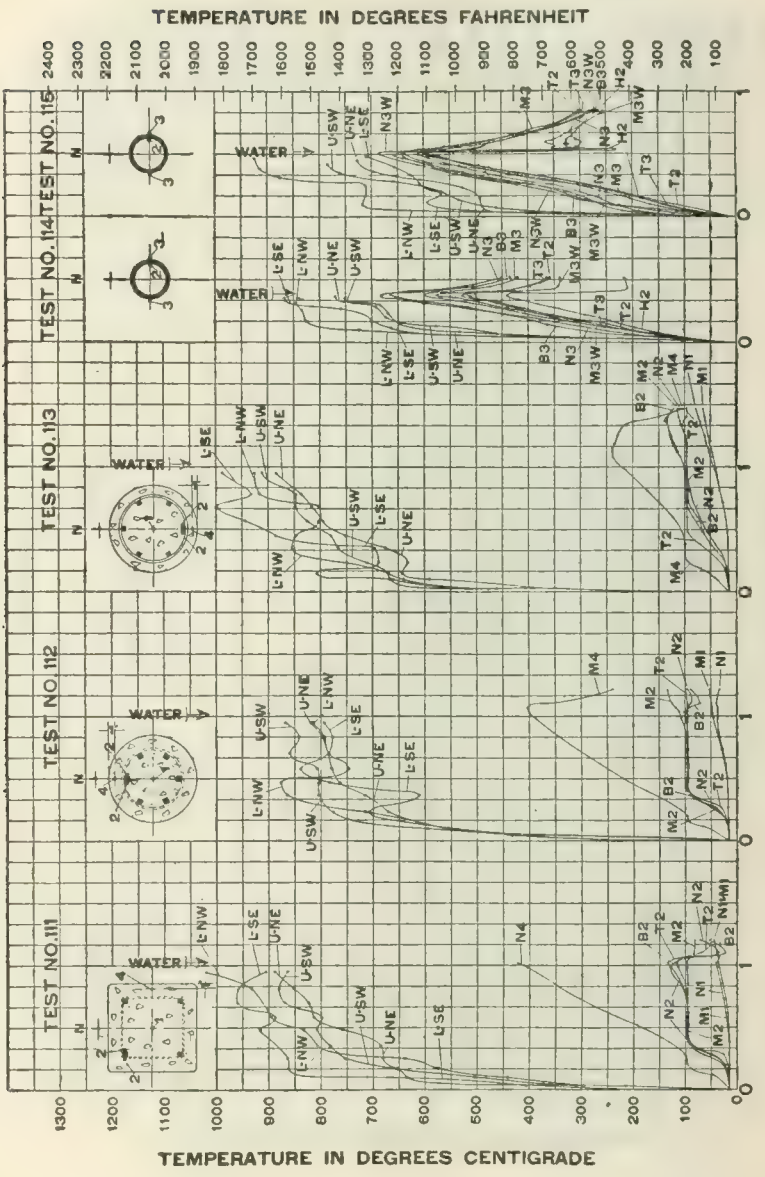


FIG. 145.—*Furnace and column temperatures, reinforced-concrete and unprotected cast-iron columns, fire and water tests, Nos. 111 to 115*

APPENDIX C.—Deformation and Average Temperature Curves

Fig. No.	Test No.	Columns and protections	Page
146	1 to 5	Unprotected structural steel	308
147	9 to 11	Unprotected cast iron	309
147	12	Unprotected pipe	309
148	23, 24	Double layer plaster on metal lath	310
149	25, 26, 27	Single layer plaster on metal lath	311
150	28A	2-inch limestone concrete	312
151	29, 36	2-inch trap-rock concrete	313
152	31, 43	2-inch sandstone concrete	314
153	32, 32A	2-inch cinder concrete	315
154	33	4-inch limestone concrete	316
155	34	4-inch granite concrete	317
156	35	4-inch limestone concrete	318
151	36	2-inch trap-rock concrete	313
157	37	4-inch trap-rock concrete	319
158	38	2-inch Joliet gravel concrete	320
159	40	2-inch trap-rock concrete	321
160	41	4-inch limestone concrete	322
161	42	do	323
152	43	2-inch sandstone concrete	314
158	47	2-inch cinder concrete	320
162	48, 49	Semifire-clay tile	324
163	50A, 51A	Surface-clay tile	325
163	52, 53	Shale tile	325
164	54, 55	Semifire-clay tile	326
165	56	do	327
165	58	Surface-clay tile	327
166	60	Semifire-clay tile	328
167	62, 63	Porous semifire-clay tile	329
166	68	2 1/4-inch surface-clay brick	328
168	69	3 3/4-inch surface-clay brick	330
169	71	Square, reinforced, trap-rock concrete	331
170	72	Round, reinforced, limestone concrete	332
171	74	Round, hooped, limestone concrete	333

NOTE.—The unit deformation is the unit longitudinal expansion or compression observed over a 37-inch (94 cm) gage length immediately above the midheight of the columns, and is given on the diagrams in terms of parts in 100 000 to the same scale as centigrade temperature. Thus a unit deformation of 100 parts in 100 000 is 0.001. Deformations above the zero line denote expansion and below compression. The initial compression shown at the beginning of each test is due to the applied working load. Cf. Section VIII paragraph *2b*, Fig. 33, and Section X, paragraph *6a*.

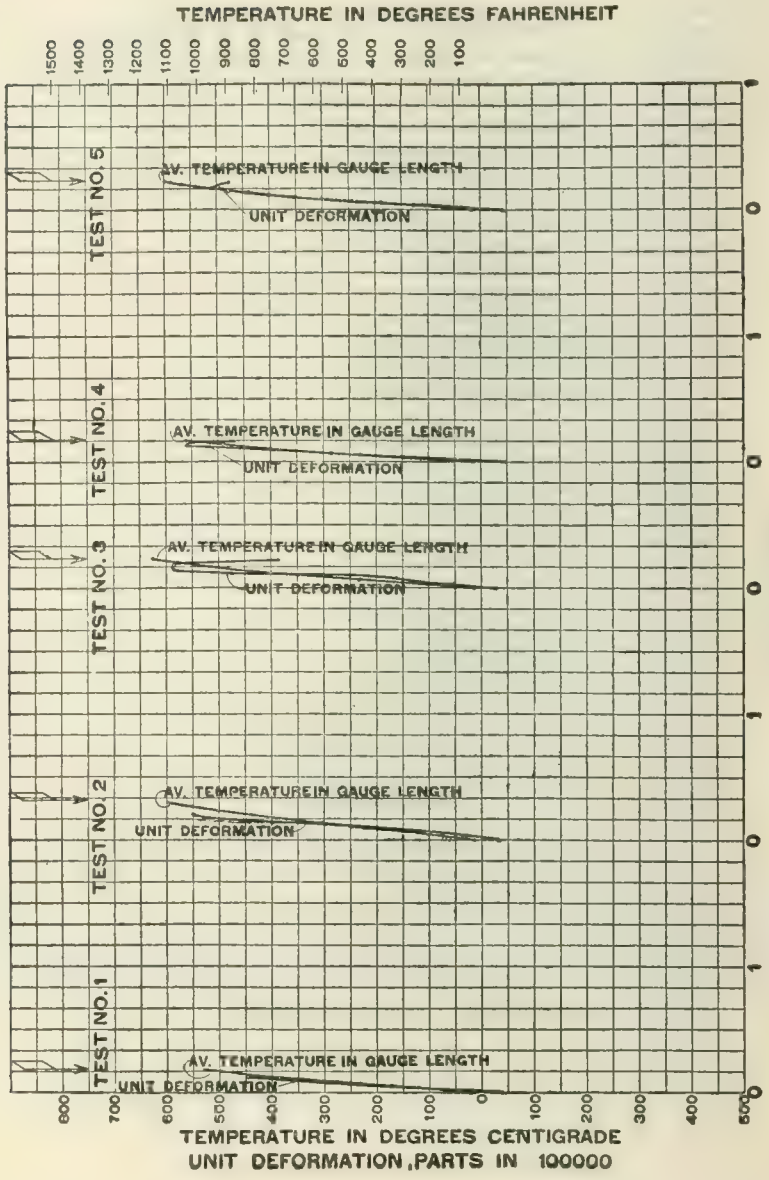

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO. 5
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 4
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 3
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 2
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 1
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TIME IN HOURS
800 700 600 500 400 300 200 100 0 100 200 300 400 500
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 146.—*Average temperature and deformation, unprotected structural-steel columns, test Nos. 1 to 5*

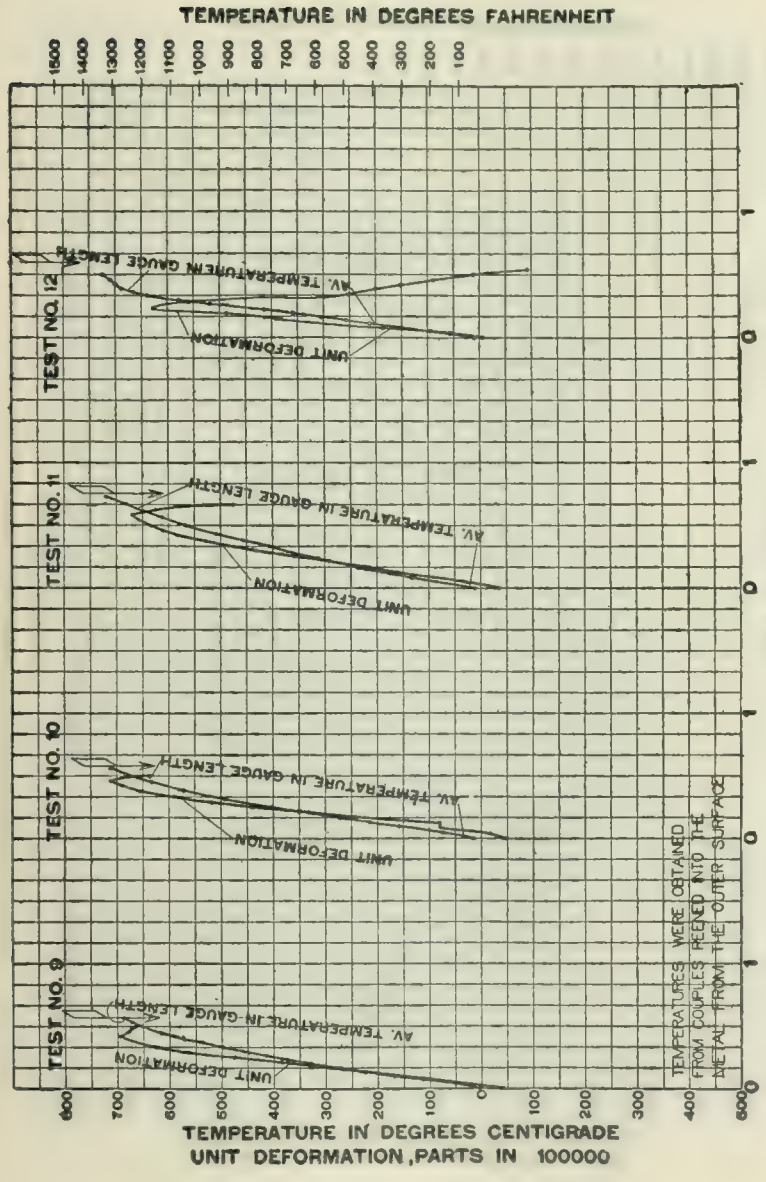

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO. 12
AV. TEMPERATURE GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 11
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 10
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 9
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TIME IN HOURS
TEMPERATURES WERE OBTAINED FROM COUPLES PEENED INTO THE METAL FROM THE OUTER SURFACE
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 147.—*Average temperature and deformation, unprotected cast-iron and pipe columns, test Nos. 9 to 12*

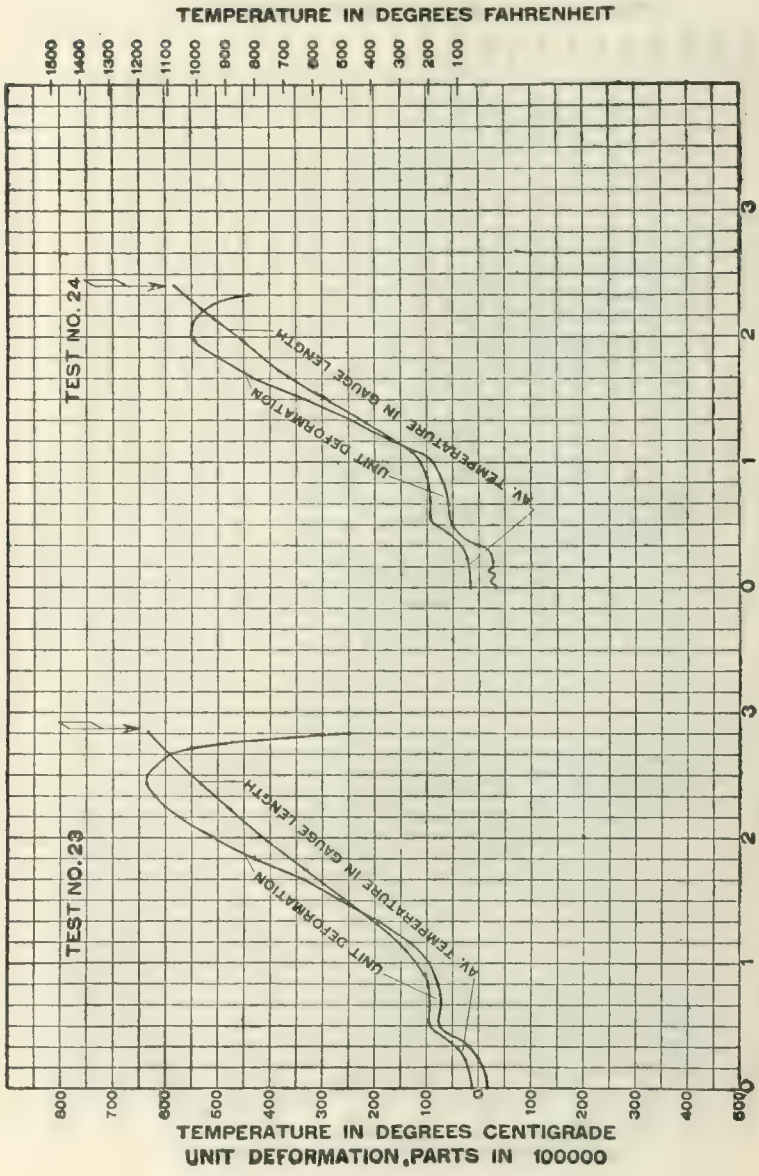

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO. 24
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 23
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TIME IN HOURS
0 1 2 3 0 1 2 3
800 700 600 500 400 300 200 100 0 100 200 300 400 600
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 148.—*Average temperature and deformation, plaster on metal-lath protections, test Nos. 23 and 24*

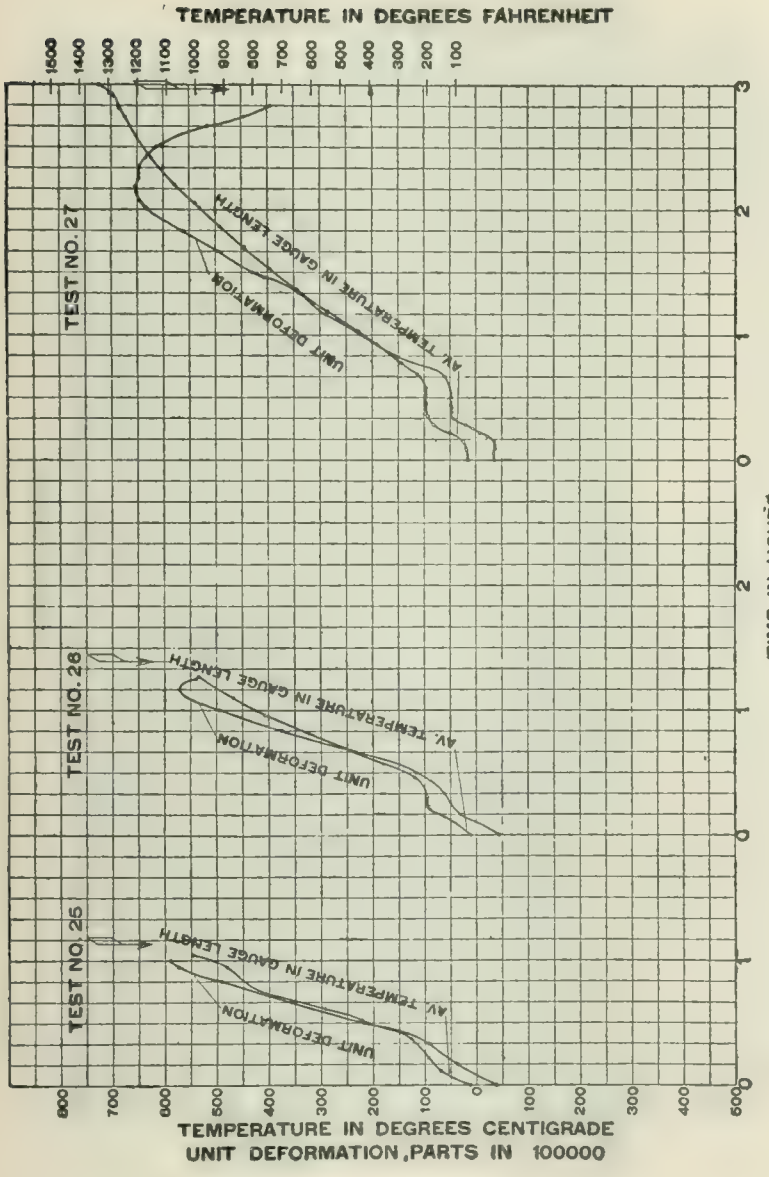

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO. 27
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 26
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 25
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
3 2 1 0 1 2 3
TIME IN HOURS
800 700 600 500 400 300 200 100 0 100 200 300 400 500
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 149.—*Average temperature and deformation, plaster on metal-lath protections, test Nos. 25, 26, and 27*

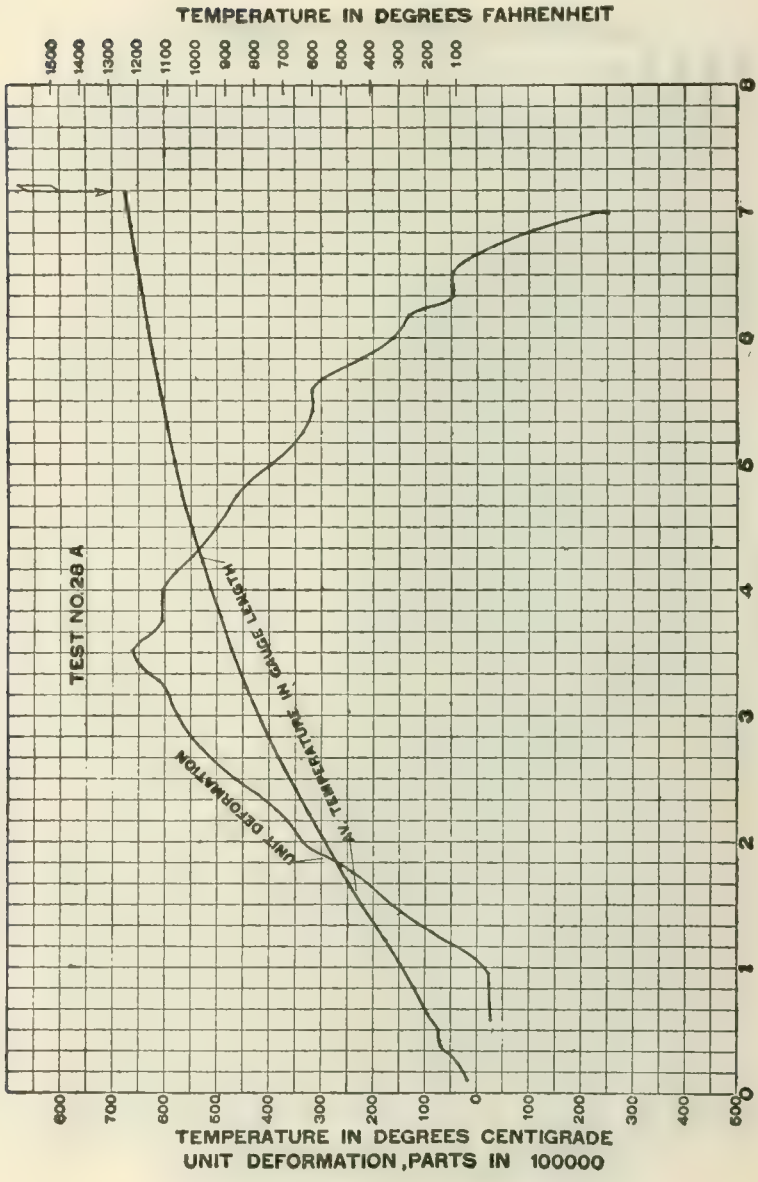


FIG. 150.—*Average temperature and deformation, 2-inch limestone concrete protection, test No. 28A*

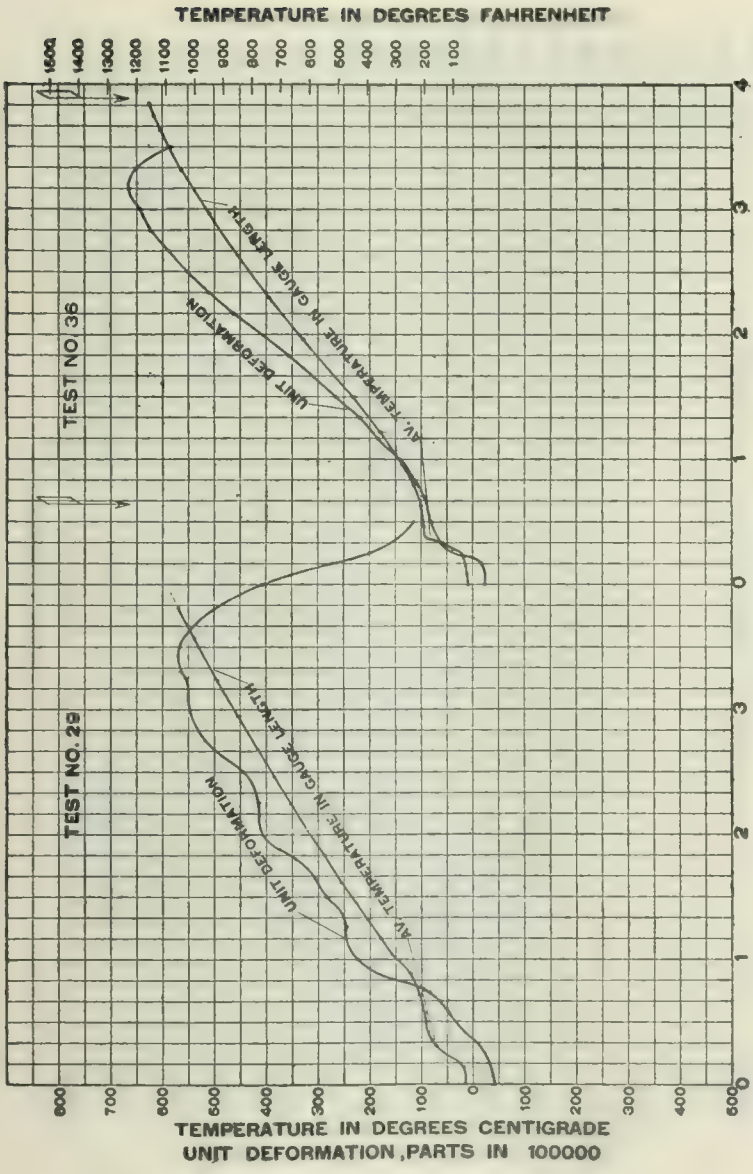


FIG. 151.—*Average temperature and deformation, 2-inch trap-rock concrete protections, test Nos. 29 and 36*

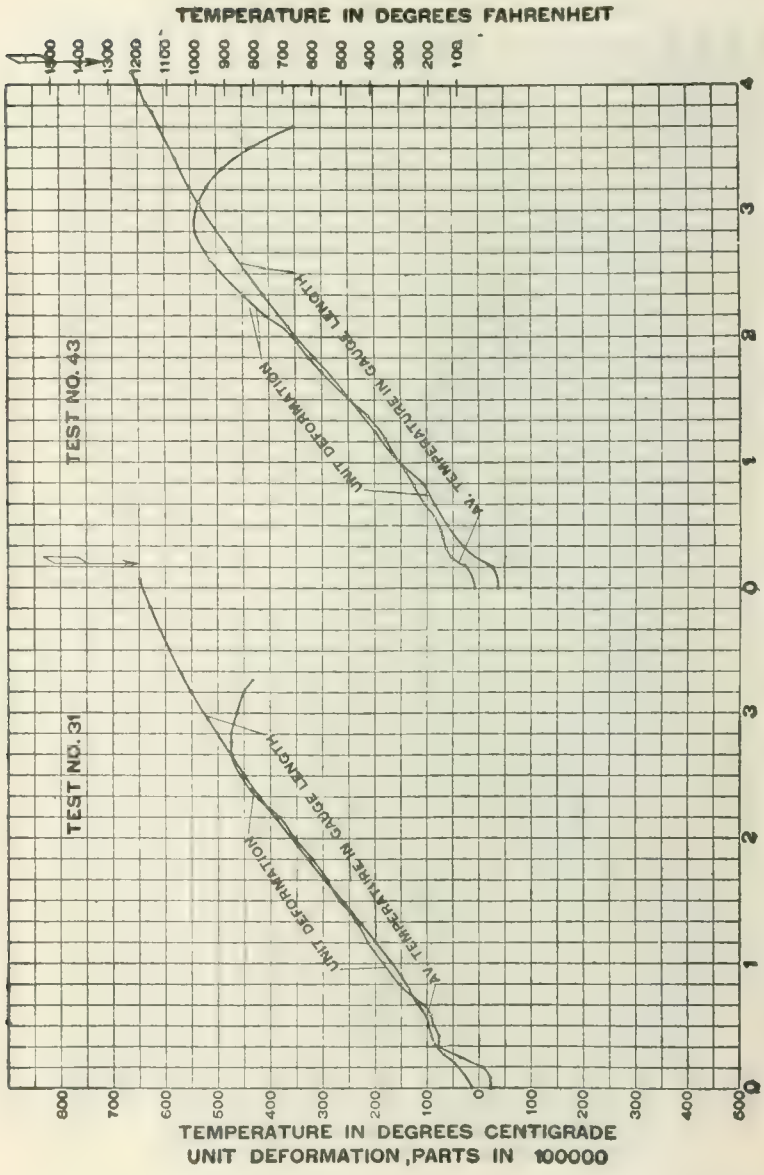


FIG. 152.—*Average temperature and deformation, 2-inch sandstone concrete protections, test Nos. 31 and 43*

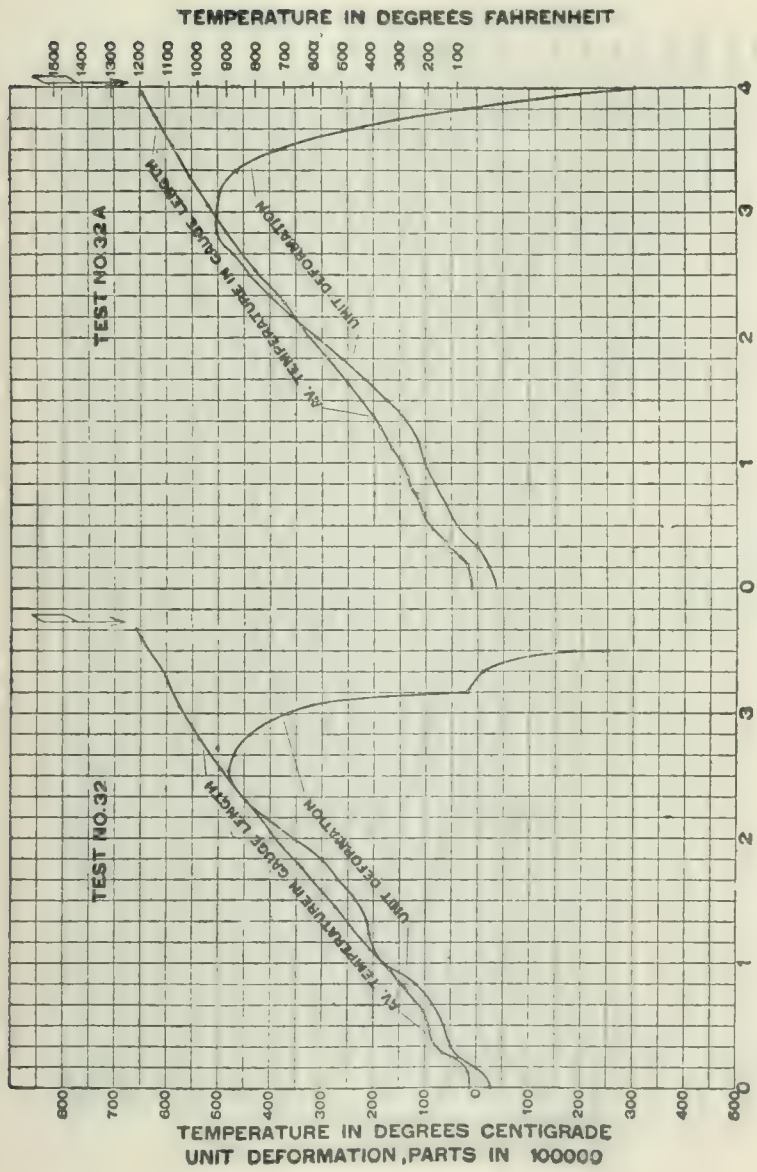


FIG. 153.—Average temperature and deformation, 2-inch cinder concrete protections, test Nos. 32 and 32A

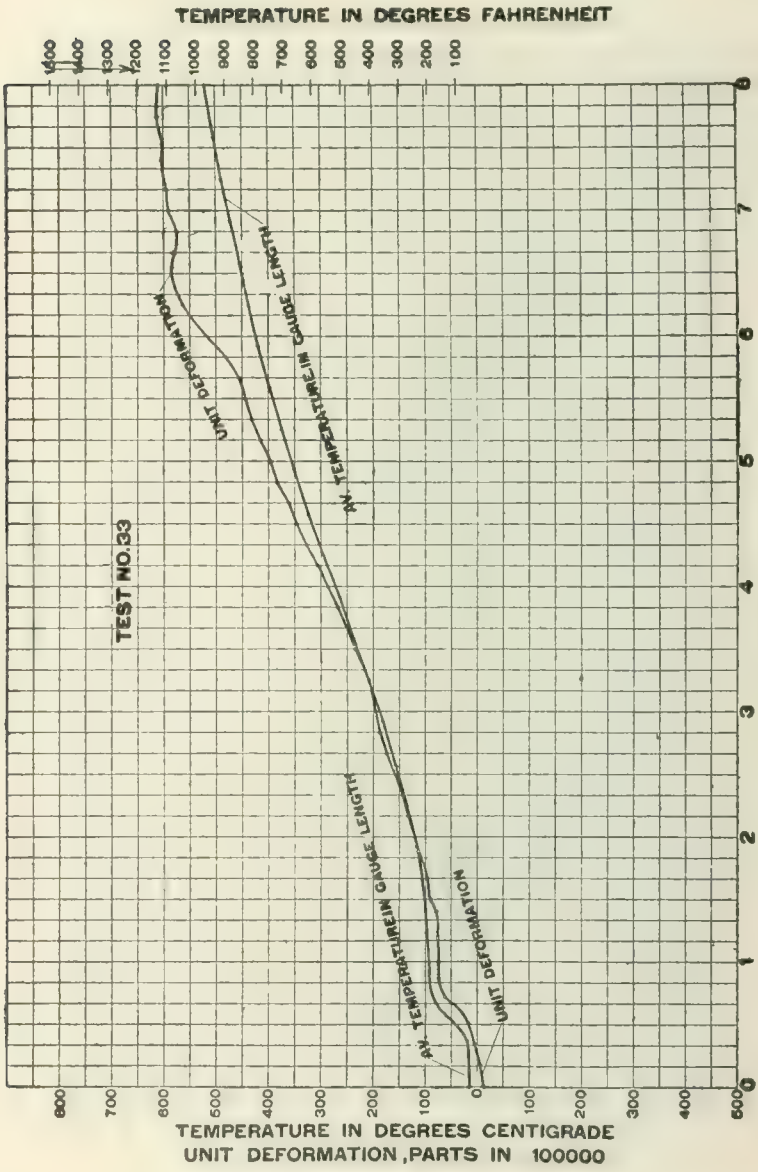


FIG. 154.—*Average temperature and deformation, 4-inch limestone concrete protection, test No. 33*

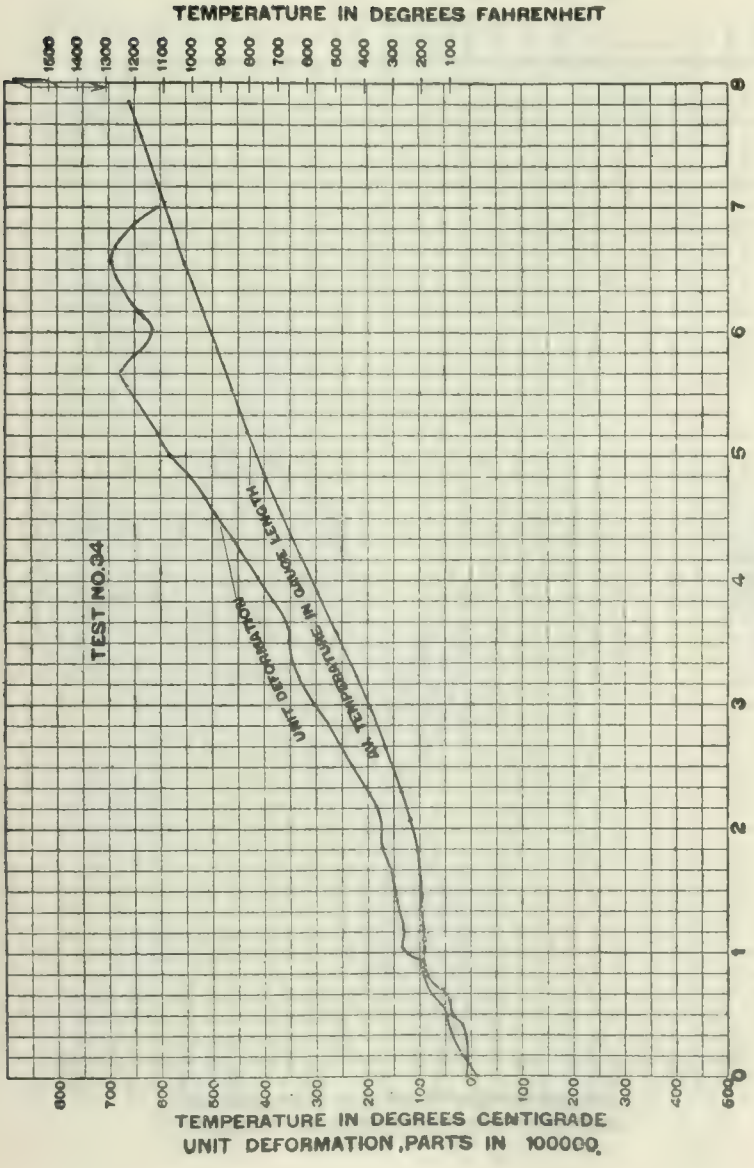


FIG. 155.—*Average temperature and deformation, 4-inch granite concrete protection, test No. 34*

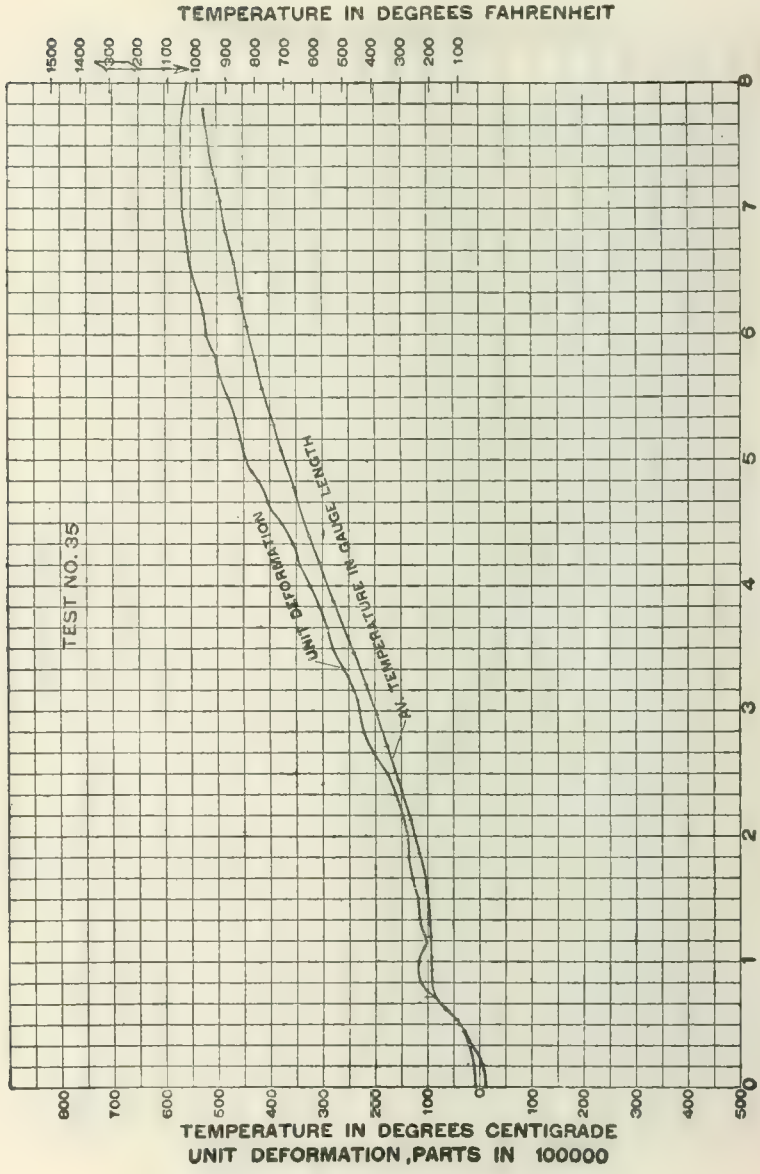


FIG. 156.—*Average temperature and deformation, 4-inch limestone concrete protection, test No. 35*

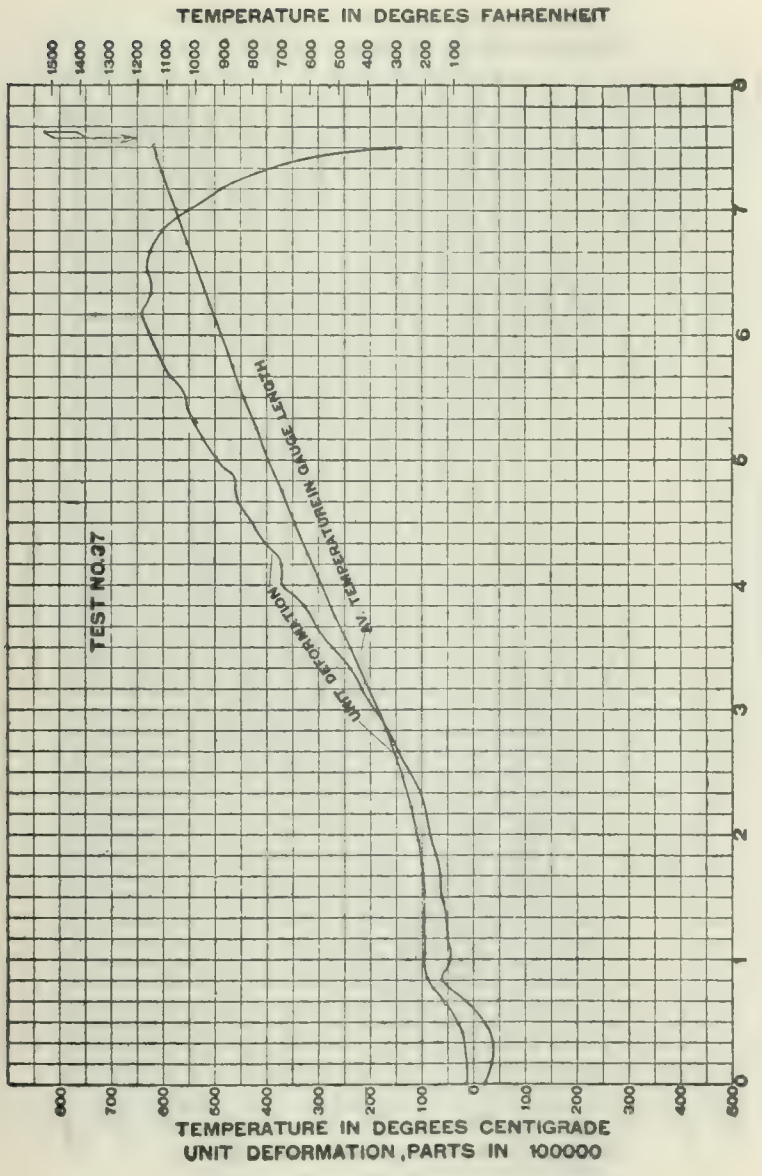


FIG. 157.—*Average temperature and deformation, 4-inch trap-rock concrete protection, test No. 37*

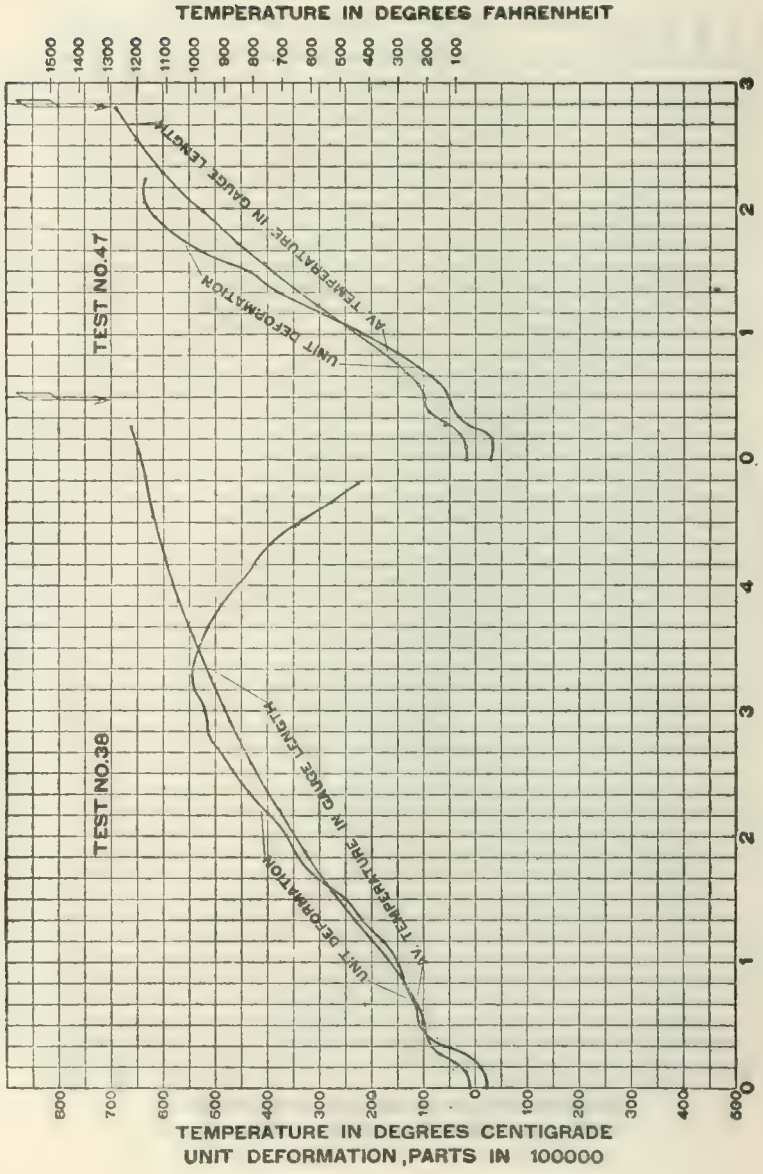

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO.47
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO.38
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TIME IN HOURS
0 1 2 3 4 3 2 1 0
800 700 600 500 400 300 200 100 0 100 200 300 400 500
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 158.—*Average temperature and deformation, 2-inch Joliet gravel concrete protection, test No. 38; 2-inch cinder concrete protection, test No. 47*

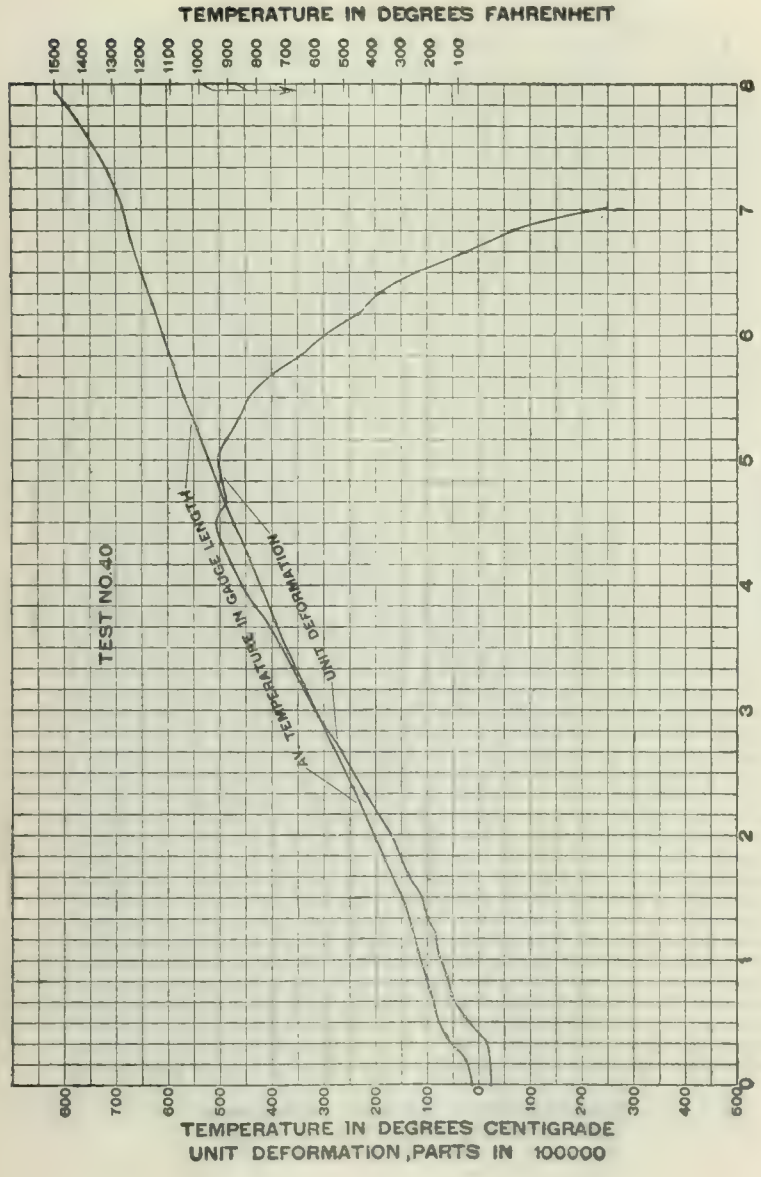


FIG. 159.—*Average temperature and deformation, 2-inch trap-rock concrete protection, test No. 40*

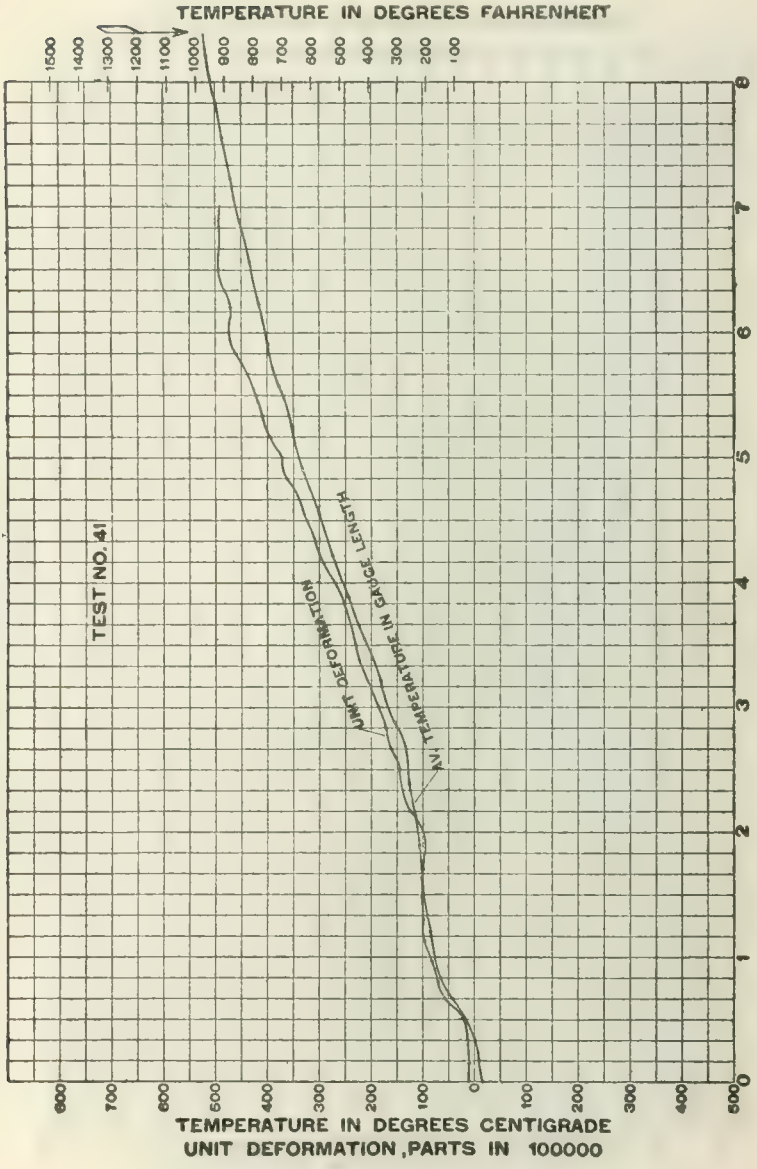


FIG. 160.—*Average temperature and deformation, 4-inch limestone concrete protection, test No. 41*

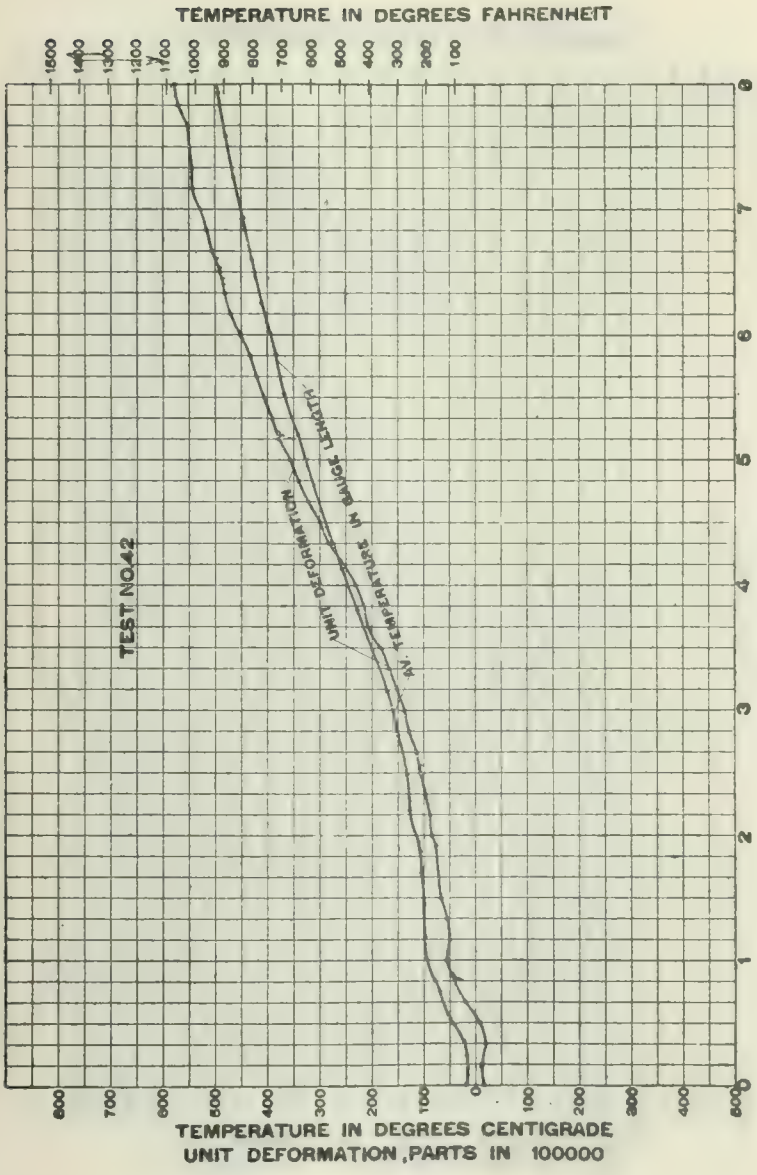


FIG. 161.—*Average temperature and deformation, 4-inch limestone concrete protection, test No. 42*

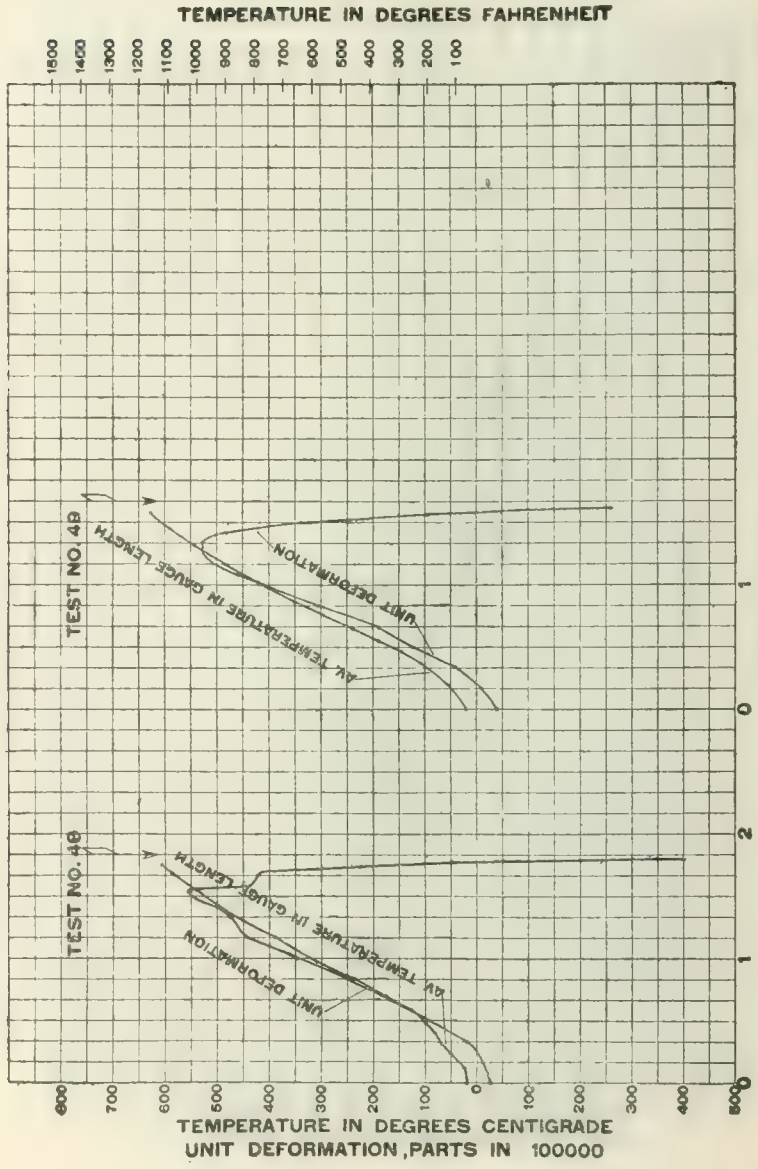


FIG. 162.—*Average temperature and deformation, semifire-clay tile protection, test Nos. 48 and 49*

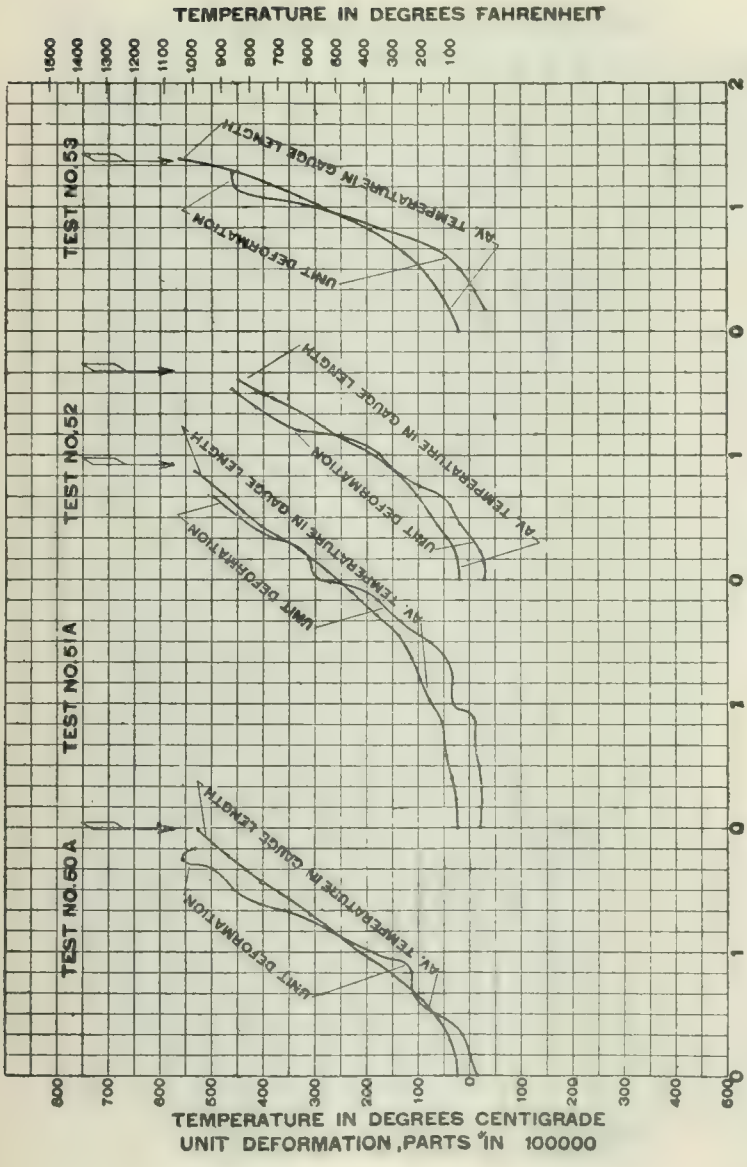

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO.53
TEST NO.52
TEST NO.51A
TEST NO.50A
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TIME IN HOURS
800 700 600 500 400 300 200 100 0 100 200 300 400 600
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 163.—*Average temperature and deformation, surface-clay tile protections, test Nos. 50A and 51A; shale tile protections, test Nos. 52 and 53*

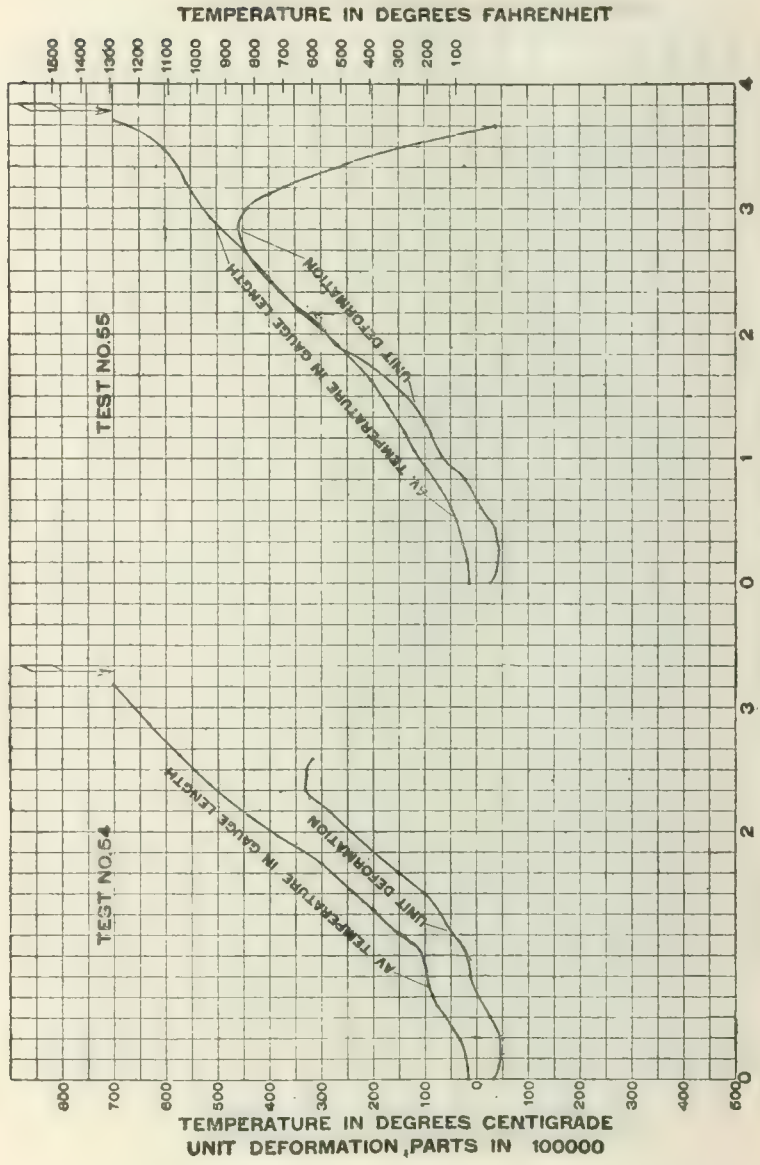


FIG. 164.—*Average temperature and deformation, semifire-clay tile protections, test Nos. 54 and 55*

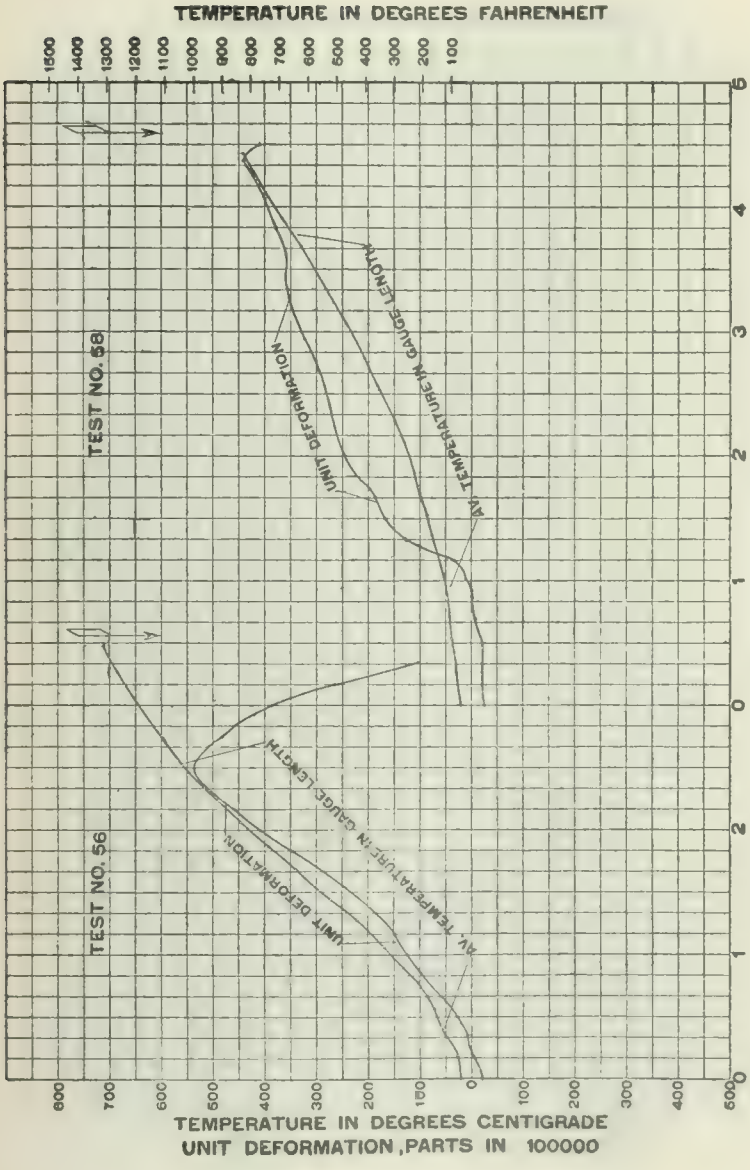


FIG. 165.—*Average temperature and deformation, semifire-clay tile protection, test No. 56; surface-clay tile protection, test No. 58*

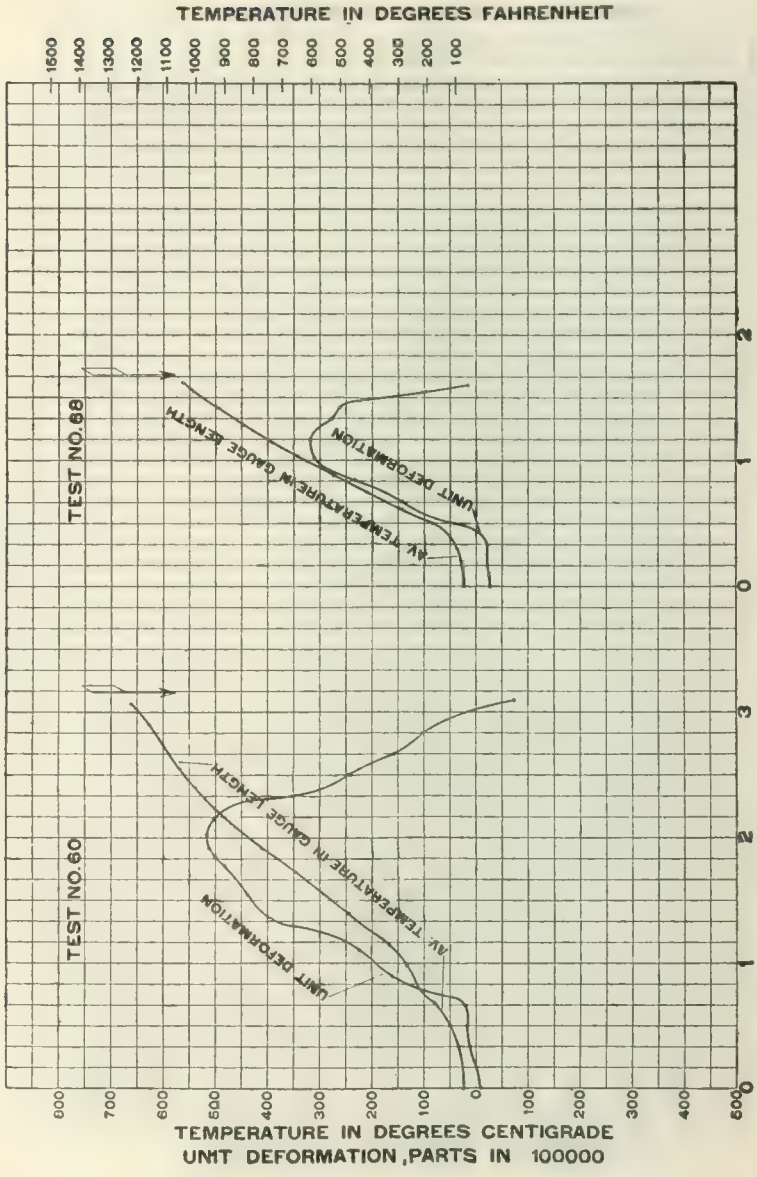


FIG. 166.—*Average temperature and deformation, semifire-clay tile protection, test No. 60; 2¼-inch surface-clay brick protection, test No. 68*

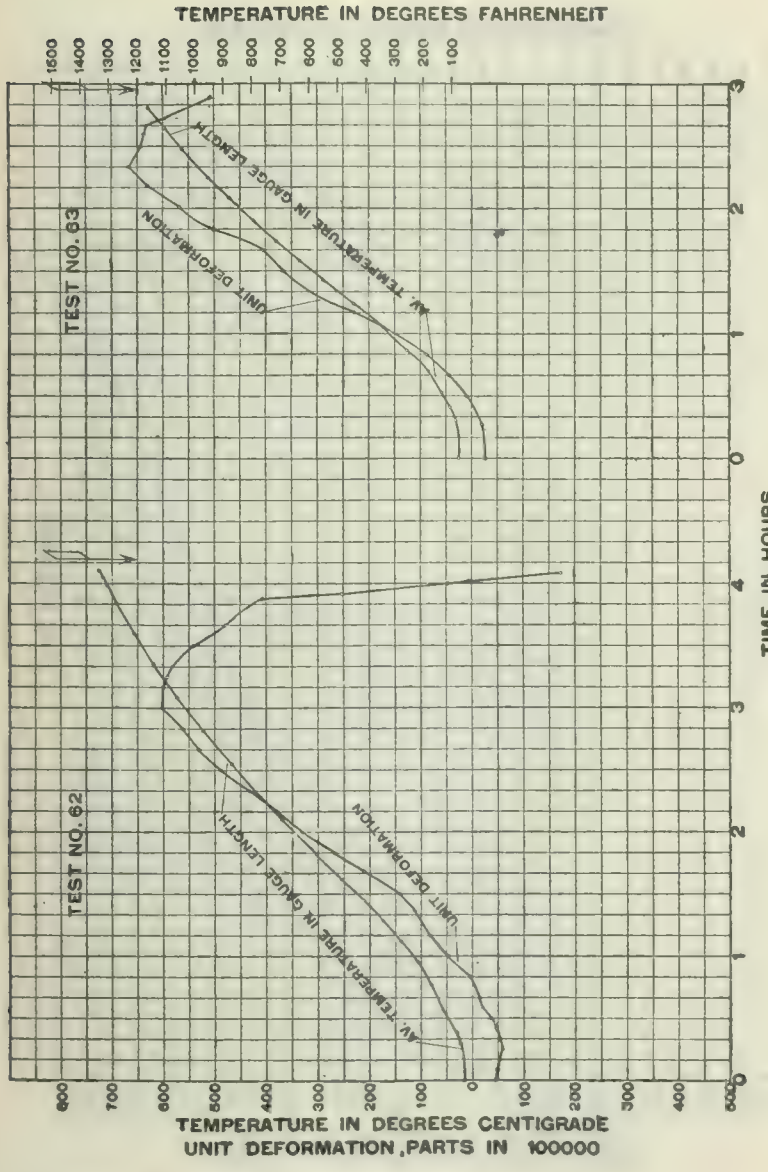

TEMPERATURE IN DEGREES FAHRENHEIT
1500 1400 1300 1200 1100 1000 900 800 700 600 500 400 300 200 100
TEST NO. 63
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TEST NO. 62
AV. TEMPERATURE IN GAUGE LENGTH
UNIT DEFORMATION
TIME IN HOURS
0 1 2 3 4 3 2 1 0
600 700 800 900 400 300 200 100 0 100 200 300 400 500
TEMPERATURE IN DEGREES CENTIGRADE
UNIT DEFORMATION, PARTS IN 100000

FIG. 167.—*Average temperature and deformation, porous semifire-clay tile protections, test Nos. 62 and 63*

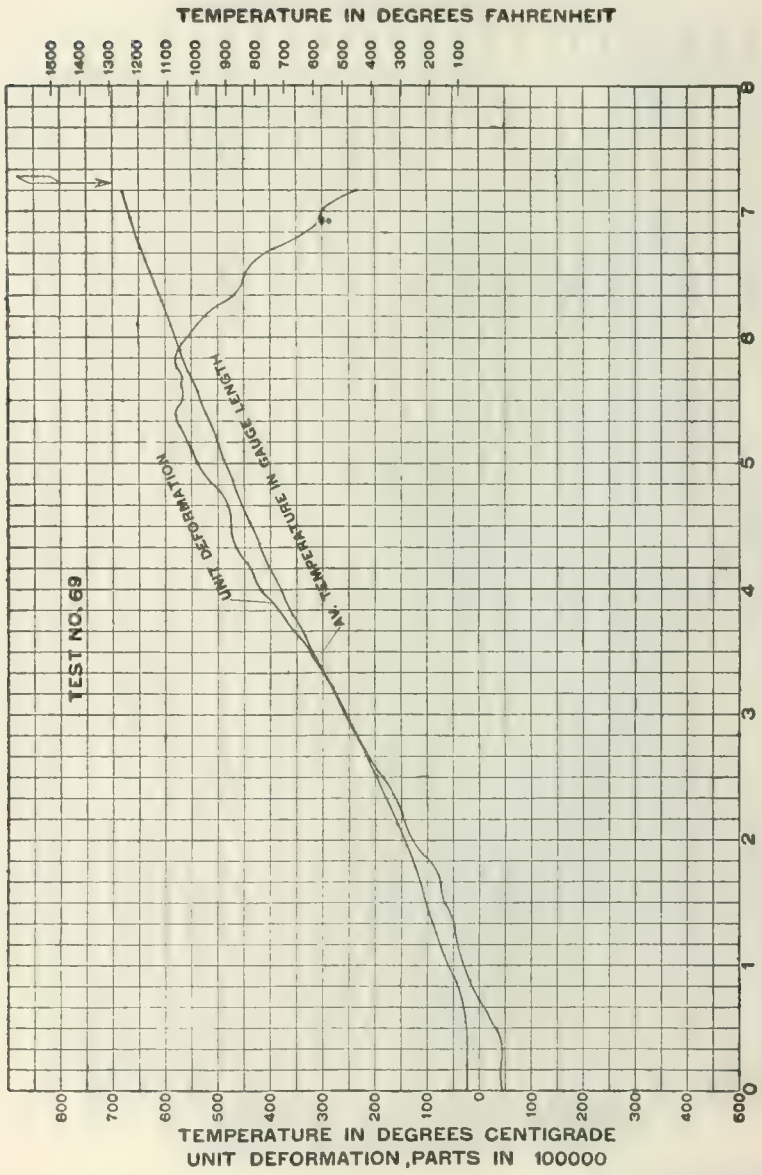


FIG. 168.—*Average temperature and deformation, 3¼-inch surface-clay brick protection, test No. 69*

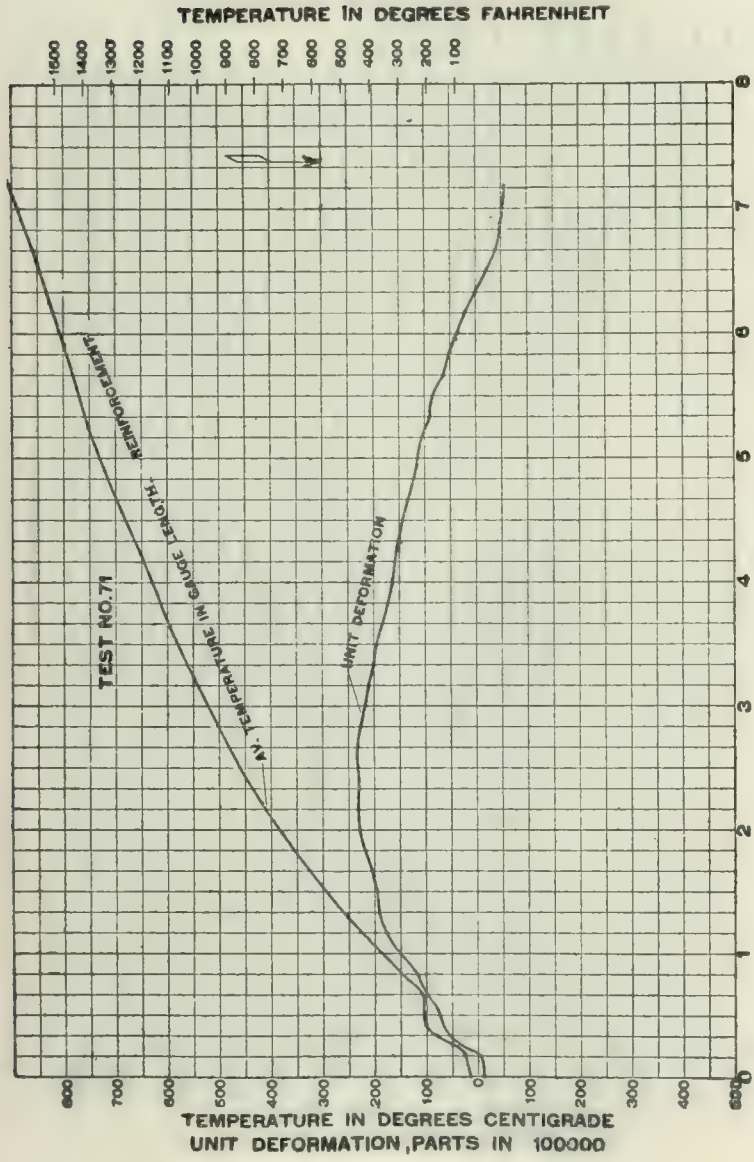


FIG. 169.—*Average temperature and deformation, square, reinforced, trap-rock concrete column, test No. 71*

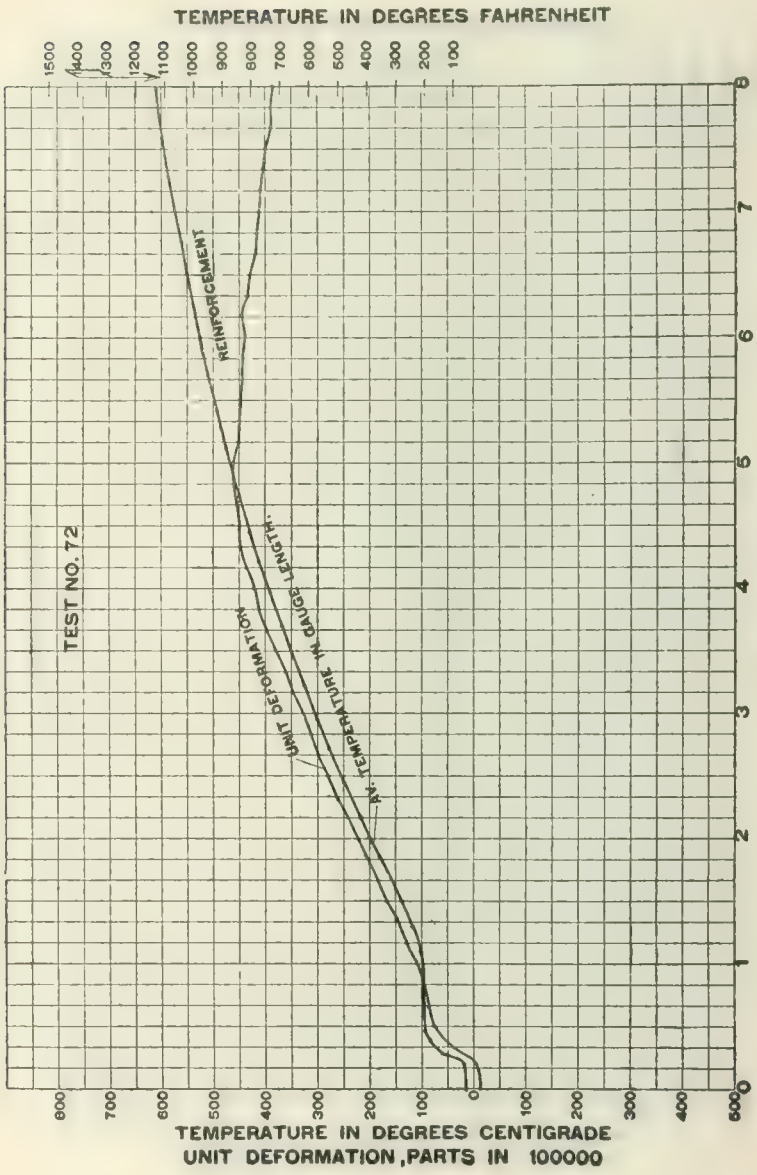


FIG. 170.—*Average temperature and deformation, round, reinforced, limestone concrete column, test No. 72*

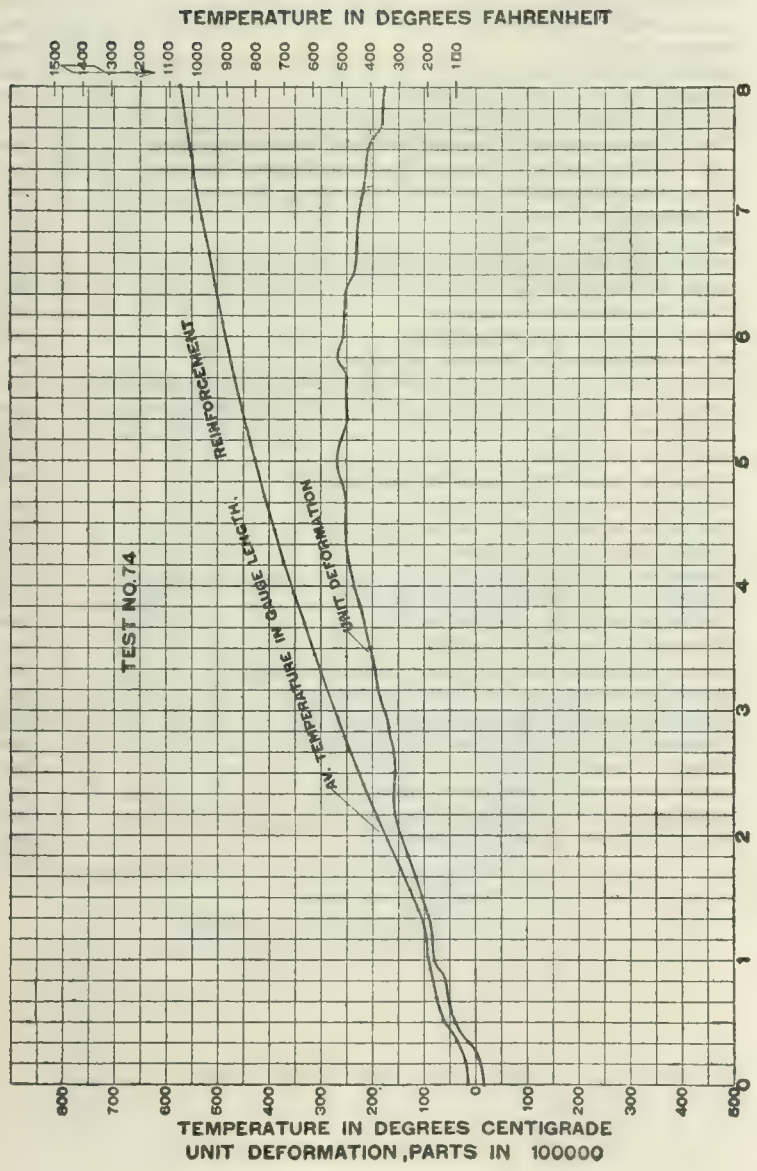


FIG. 171.—*Average temperature and deformation, round, hooped, limestone concrete column, test No. 74*

APPENDIX D.—Tables of Auxiliary Tests of Materials

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NOTE.—For details of tests see Section V, pp. 58–74.

TABLE 5.—TENSION TESTS OF STEEL

Column Test No.	Specimen No.	SECTION	Location of Specimen	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per sq. In.	Elongation in 2 In., Percent	Reduction in Area, Percent
1	T-1	Rolled H	Edge of flange	40025	62600	41.0	57.8
1	T-1A	do	Junction of web and flange	37500	60000	37.0	61.5
1	T-1B	do	Center of web	42300	62600	40.0	52.8
14	T-14	do	do	.....	76300	32.0	61.0
28	T-28	do	do	34850	57400	37.5	65.6
28A	T-28A	do	do	37500	68250	32.5	59.2
30	T-30	do	do	34100	57600	25.0	67.0
31	T-31	do	do	38000	58600	29.0	60.0
32	T-32	do	do	43900	69400	17.5	61.0
33	T-33	do	do	35000	59000	27.5	66.0
33A	T-33A	do	do	37200	71500	24.0	54.0
34	T-34	do	do	48050	69800	25.0	55.0
34A	T-34A	do	do	32200	62700	21.0	66.0
35	T-35	do	do	34000	66000	16.0	63.0
48	T-48	do	do	39200	60800	25.0	61.0
49	T-49	do	Edge of flange	42900	64700	40.0	60.3
49	T-49A	do	Junction of web and flange	37750	60400	40.0	68.5
49	T-49B	do	Center of web	44700	62300	35.0	54.2
67A	T-67A	do	Edge of flange	41260	62230	43.0	62.8
68	T-68	do	do	35300	56720	47.0	64.3
16	T-16	Plate and Angle	Edge of angle	42850	59800	27.0	62.0
23	T-23	do	Center of plate	35500	58900	28.5	56.0
36	T-36	do	do	32100	56600	26.5	65.0
50A	T-50A	do	do	37650	60200	28.0	53.0
51	T-51	do	Edge of angle	34200	60500	25.0	61.0
105	T-105	do	Center of plate	29450	55300	27.5	69.0
24	T-24	Plate and Channel	Center of channel web	.....	61300	31.0	58.0
38	T-38	do	do	40500	67600	23.0	54.0
39	T-39	do	do	36600	58900	32.0	66.0
53	T-53	do	do	39000	62500	26.0	57.0
65	T-65	do	Edge of channel flange	37850	57680	39.0	58.5
65	T-65-1	do	Junction of flange and web	.....	51600	34.0	43.1
107	T-107	do	Center of channel web	46300	63400	27.0	54.0

TABLE 5.—TENSION TESTS OF STEEL—Continued

Column Test No.	Specimen No.	SECTION	Location of Specimen	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation in 2 In., Percent	Reduction in Area, Percent
18	T-18	Latticed Channel	Center channel web	44400	70900	23.0	55.0
40	T-40	do	do	40500	62200	28.0	63.0
19	T-19	Z-Bar and Plate	Center of web	35700	60500	25.0	55.0
25	T-25	do	do	32900	52200	28.0	65.0
32A	T-32A	do	do	35050	57200	30.5	79.0
41	T-41	do	do	31000	54400	23.0	62.0
42	T-42	do	do	44300	69700	17.5	58.0
55	T-55	do	do	43000	65000	19.0	55.0
56	T-56	do	do	37100	54200	25.0	63.0
20	T-20	I-Beam and Channel	Channel web	44800	64800	26.0	55.0
21	T-21	do	do	40500	62200	28.0	52.0
57	T-57	do	do	42100	65300	24.0	56.0
59	T-59	do	do	44000	63400	19.0	55.0
22	T-22	Latticed Angle	Edge of angle	42900	63700	18.0	63.0
46	T-46	do	do	47800	66300	35.0	64.0
8	T-8	Star	Edge of angle	39600	69300	24.0	62.0
45	T-45	do	do	43200	61100	26.5	64.0
70	70	Vertically Reinforced Concrete.	1 in. sq. deformed reinforcing bar	*34200	59500	22.8	63.4
70	70A	do	do	27340	47860	30.3	66.7
72	72	do	do	*36000	54000	26.8 (a)	62.0
72	72A	do	do	27420	46600	31.8	67.3
73	73-1	do	do	*33500	56000	27.0	62.5
73	73-1A	do	do	*35000	55750	25.8	64.5
73	73-2	do	do	*27730	46650	33.8	68.3
111	111-1	do	do	*35000	56000	24.2 (a)	64.5
111	111-1A	do	do	*35000	56500	26.5 (a)	64.5
111	111-2	do	do	35100	52570	32.5	67.8
112	112-1	do	do	*51000	68500	22.5	62.0
112	112-1A	do	do	*50000	65000	21.9	62.0
112	112-2	do	do	20870	34750	41.0	77.2

(a) Elongation was measured in 8-in. gage length instead of 2 in.
*Tested on reduced machined sections with shouldered ends.

TABLE 5.—TENSION TESTS OF STEEL—Concluded

Column Test No.	Specimen No.	SECTION	Location of Specimen	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation in 2 In., Percent	Reduction in Area, Percent
74	74 1	Hooped Reinforced Concrete	¾ in. sq. deformed reinforcing bar	*40000	57500	28. 3	64. 2
74	74-1A	do	do	*36820	56700	31. 0	64. 6
74	74-2	do	do	36870	51680	35. 8	67. 7
74	74-3	do	do	*38800	54700	29. 3	68. 9
74	74-3A	do	do	40000	53500	31. 0	64. 7
74	74-4	do	do	*40850	59350	21. 8	60. 5
74	74-4A	do	do	*42300	59200	30. 3	66. 0
75	75-1	do	do	*37800	57700	29. 5	64. 4
75	75-1A	do	do	*39000	57500	29. 8	65. 8
75	75-2	do	do	37750	52870	35. 8	64. 6
113	113-1	do	do	*39800	58700	29. 3	63. 2
113	113-1A	do	do	*39800	59200	30. 3	63. 2
113	113-2	do	do	37950	54950	31. 3	64. 8
	Q-1	do	¼ in. high carbon wire hooping	55500	86900	....	....
	Q-2	do	do	49000	87600	....	....
...	Q-3	do	do	54000	87600	....	....
..	K 1	Ties in partly protected columns	½ in. plain round bar	36680	58700	29. 0	68. 4
...	K 2	do	do	33930	54600	33. 0	66. 2
...	K-3	do	do	33000	48200	30. 3	72. 2
...	L-1	do	⅜ in. plain round bar	40420	57700	27. 5	68. 9
...	L-2	do	do	40230	56950	27. 3	70. 5
...	L-3	do	do	34400	56170	28. 5	70. 7
...	M-1	Ties in vertically reinforced concrete columns	¼ in. plain round bar	41150	58500	....	63. 5
...	M-2	do	do	41410	60000	....	67. 4
...	M-3	do	do	41000	62500	....	67. 8
...	M-4	do	do	44000	59100	35. 0	70. 4
...	M-5	do	do	43600	61200	36. 0	68. 6
...	M-6	do	do	43050	57500	38. 0	71. 6
...	N-1	Ties in concrete protections	No. 5 (B. & S. ga.) bright basic wire	.....	76200	....	67. 5
...	N-2	do	do	.....	73800	....	67. 5
...	N 3	do	do	.....	75500	....	67. 5

* Tested on reduced machined sections with shouldered ends.

TABLE 6.—TENSION TESTS OF METAL LATH

Each value is the average of three tests

Specimen No.	Kind of Lath	Width of Specimen	Ultimate Strength, Lb. per In. Width
R-1	No. 24 Gage Expanded Metal	3 in.	150
R-2	No. 24 High-Ribbed Metal Lath	2½ in.	520
R-3	Woven Wire Lath	3 in. (7 strands)	256

TABLE 7.—HARDNESS TESTS OF STEEL

Column Test No.	Specimen No.	Section	Location of Specimen	Brinnell Hardness No.
2	H-2	Plate and Angle	Edge of angle	113
17	E-17	do	do	121
37	E-37	do	do	118
51A	E-51A	do	do	118
103	H-103	do	do	121
104	H-104	do	do	125
106	H-106	do	do	131
110	H-110	do	do	126
60	H-60	Latticed angle	do	123
				Scleroscope Hardness No.
3	HC-3	Plate and Channel	Center of channel web	23
3	HP-3	do	Plate	24
38	H-38	do	do	21
107	H-107	do	Center of channel web	24
66	H-66	Latticed channel	Center of web	23
5	H-5	Z-bar and Plate	Plate	26
41	H-41	do	do	22
56	H-56	do	do	24
6	H-6	I-beam and Channel	Channel web	26
43	H-43	do	Center of I-beam web	22
57	H-57	do	do	22
58	H-58	do	do	22
7	H-7	Latticed angle	Edge of angle	25
26	H-26	do	do	20
60	H-60	do	do	30
61	H-61	do	do	22

TABLE 8.—CHEMICAL ANALYSES OF STEEL

Column Test No.	Specimen No.	Carbon	Manganese	Phosphorus	Sulphur	Silicon
1	1	0. 20	0. 54	0. 010	0. 021	0. 03
1	1A	0. 18	0. 54	0. 015	0. 028	0. 01
1	1B	0. 21	0. 54	0. 012	0. 025	0. 04
14	14	0. 16	0. 72	0. 023	0. 034	0. 03
65	65	0. 18	0. 40	0. 011	0. 036	0. 04
65	65–1	0. 20	0. 41	0. 010	0. 038	0. 05
65	65–2	0. 21	0. 42	0. 006	0. 029	0. 04
68	68	0. 12	0. 57	0. 010	0. 022	0. 02
22	Rivets	0. 09	....	0. 007	0. 050	....
46	Rivets	0. 11	....	0. 024	0. 064	....

TABLE 9.—TRANSVERSE AND TENSION TESTS OF CAST IRON

Column Test No.	Specimen No.	Modulus of Rupture, Lb. per Sq. In.	*Ultimate Tensile Strength, Lb. per Sq. In.
9	T–9	47800	23230
10	T–10	45800	29470
11	T–11	43500	23900
27	T–27	64700	26700
47	T–47	.....	23050
62	T–62	56100	28800
**10A	1CIA	.....	17600
114 and	1CIB	39400	18300
115	1CIC	36200	17100
do	2CIA	45500	20300
do	2CIB	46300	21700
do	2CIC	45700	21600

*Tension specimens cut from ends of transverse specimens after test.
**Specimens cut from duplicate test column.

TABLE 10.—CHEMICAL ANALYSIS OF CAST IRON

Column Test No.	Specimen No.	Carbon	Manganese	Phosphorus	Sulphur	Silicon	Graphite
9	T–9	0. 61	0. 47	0. 57	0. 090	1. 86	2. 66
10	T–10	0. 67	0. 50	0. 57	0. 090	1. 86	2. 77
11	T–11	....	....	....	0. 087	....	....
27	T–27	....	....	....	0. 086	....	....
62	T–62	....	....	....	0. 082	....	....

TABLE 11.—TESTS OF PORTLAND CEMENT

By Bureau of Standards' Washington Laboratory

DATE OF REPORT	September 22, 1916							Dec. 19, 1916
SAMPLE No.	1	2	3	4	5	B-1	B-2	12*
Chemical Analysis:								
Silica (SiO_2)	20.34%	19.46%	20.36%	20.28%	20.40%	20.46%	20.82%	20.82%
Oxide of Iron (Fe_2O_3)	2.72%	2.70%	2.60%	2.70%	2.80%	2.75%	2.52%	2.10%
Oxide of Aluminum (Al_2O_3)	6.98%	7.18%	7.24%	6.86%	6.74%	6.15%	6.38%	7.14%
Lime (CaO)	63.68%	62.12%	63.42%	64.32%	63.24%	62.86%	64.68%	59.50%
Magnesia (MgO)	1.13%	1.21%	0.90%	1.18%	1.12%	1.22%	1.28%	3.46%
Sulphuric Anhydride (SO_3)	1.34%	1.44%	1.45%	1.45%	1.46%	1.38%	1.45%	1.50%
Loss of Ignition	4.53%	3.43%	3.17%	3.17%	3.22%	3.40%	2.26%	4.48%
Insoluble Residue	0.10%	0.13%	0.17%	0.20%	0.13%	0.16%	0.13%	0.70%
Specific Gravity:								
As Received	3.06	3.09	3.08	3.08	3.08	3.10	3.12	.....
After Ignition	3.17	3.18	3.16	3.22	3.15	.....	.....	3.17
Fineness:								
Passing 110-Mesh Sieve	94.8%	94.4%	97.0%	96.0%	97.0%	97.4%	96.4%	99.2%
Passing 200-Mesh Sieve	77.6%	76.0%	77.4%	78.4%	77.0%	78.5%	78.1%	75.0%
Tensile Strength, Lb. per Sq. In.:								
Neat, 7 Days	434	552	551	587	680	622	736	435
	471	512	641	645	527	564	575	510
	575	610	562	604	577	670	587	425
Neat, 28 Days	730	690	670	690	545	715	725	740
	635	655	710	640	670	800	675	625
	585	595	730	575	705	700	665	675
1 Cement: 3 Sand—								
7 Days	178	227	222	182	228	121	193	180
	178	229	182	207	229	177	247	140
	169	206	206	213	243	209	238	165
28 Days	280	295	325	325	330	265	365	375
	310	300	310	335	340	280	380	380
	260	265	310	330	290	325	330	340
Time of Set, hr.—min.:								
Initial	7-30	7-15	7-30	7-10	7-20	7-30	7-25	7-0
Final	11-50	11-30	11-55	11-45	11-40	11-50	11-00	10-30
Soundness: Pats—								
28 Days in Air	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O K.
28 Days in Water	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.
5 Hr. in Steam	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.
Water Used:								
Neat	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	23.0%
1:3 Mortar	10.7%	10.7%	10.7%	10.7%	10.7%	10.7%	10.7%	10.3%

*Portland cement used in pipe column, Test No. 12.

TABLE 12.—TESTS OF PORTLAND CEMENT

By Bureau of Standards' Pittsburg Laboratory

DATE OF REPORT	May 1, 1917			June 19, 1917		Nov. 19, 1918	
SAMPLE No.	H-1	H-2	H-3	H-5	H-6	H-7	H-8
Specific Gravity:							
As Received	3.15	3.18	3.18	3.10	3.11	3.11	3.12
After Ignition	3.16	3.18	3.19	.....	.....	.....	.....
Fineness:							
Passing 100-Mesh Sieve	95.0%	96.0%	95.4%	94.0%	94.2%	.....	.....
Passing 200-Mesh Sieve	77.2%	77.7%	77.0%	76.2%	76.4%	78.8%	78.6%
Tensile Strength, Lb. per Sq. In.:							
Neat, 7 Days	516	598	695	621	450	.....	.....
	555	550	685	474	521	.....	.....
	521	510	650	530	543	.....	.....
Neat, 28 Days	725	750	700	585	661	.....	.....
	745	688	690	600	575	.....	.....
	650	681	776	570	691	.....	.....
1 Cement: 3 Sand—							
7 Days	190	204	261	203	205	300	310
	208	242	260	225	225	260	300
	245	223	269	212	239	300	305
28 Days	358	282	322	315	349	410	440
	371	332	326	331	381	430	470
	376	345	340	382	352	400	460
Time of Set, hr.—min.:							
Initial	7–0	7–0	7–0	Over 1 hr.	Over 1 hr.	3–50	3–50
Final	Over 10 hrs.	Over 10 hrs.	Over 10 hrs.	Over 10 hrs.	Over 10 hrs.	8–50	9–0
Soundness: Pats—							
5 Hr. in Steam	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.
Water Used:							
Neat	24.0%	24.0%	24.0%	24.0%	24.0%	.....	.....
1:3 Mortar	10.5%	10.5%	10.5%	10.5%	10.5%	.....	.....

TABLE 13.—TESTS OF PORTLAND CEMENT

By R. W. Hunt & Company

DATE OF REPORT	May 2, 1917		
SAMPLE No.	H-1	H-2	H-3
Specific Gravity:			
As Received	3.14	3.13	3.13
Fineness:			
Passing 100-Mesh Sieve	95.4%	96.0%	96.1%
Passing 200-Mesh Sieve	76.4%	78.0%	77.2%
Tensile Strength, Lb. per Sq. In.:			
Neat, 24 Hr.	305	355	345
	320	330	330
	335	350	300
Neat, 7 days	690	625	675
	640	640	650
	700	700	695
Neat, 28 Days	700	700	695
	730	765	710
	765	725	725
1 Cement, 3 Sand:			
7 Days	310	330	300
	330	310	320
	335	345	335
28 Days	410	385	410
	380	395	400
	405	410	385
Time of Set, hr.—min.:			
Initial	5–45	5–40	5–40
Final	8–05	8–0	8–10
Soundness: Pats—			
5 Hr. in Steam	O. K.	O. K.	O. K.
Water Used:			
Neat	23.0%	23.0%	23.0%
1 : 3 Mortar	10.3%	10.3%	10.3%

TABLE 14.—CHEMICAL ANALYSES OF SANDS

SAND	Insoluble in Acid	Al_2O_3 and Fe_2O_3	CaO	MgO	CO_2	*Loss on Ignition
Fox River	54.51	1.34	14.90	8.24	21.22	.....
Joliet	52.10	1.19	15.46	8.97	23.06	.....
Meramec River	98.66	0.58	0.14	0.06	0.11	.....
Long Island	97.98	0.96	0.04	0.18	0.25	.....
Pelee Island	64.70	1.66	17.03	1.34	15.68	.....
Plum Island	98.18	0.66	0.16	0.12	0.18	.....
Cambridge	96.93	1.63	0.17	0.90	0.25	0.97
Coarse lake	83.53	6.69	3.31	1.21	3.30	3.63
Fine lake	87.69	5.38	2.17	0.93	1.95	2.28
Beach	85.45	5.60	2.35	1.02	2.29	2.79
Bank	91.30	5.18	0.09	0.51	0.22	0.54

*Not including CO_2.

TABLE 15.—PHYSICAL PROPERTIES OF CONCRETE SANDS

SAND.	Specific Gravity	Weight per Cu. Ft. (Dry), Lb.	Computed Voids, Per cent	Granular Analysis—Percent Passing Sieve No.											
				3	4	6	8	10	14	20	28	35	48	65	100
Fox River	2.62	110.0	33.0	99.4	93.5	86.9	79.6	71.4	63.7	53.2	37.1	22.9	11.0	5.3	2.5
Joliet	2.62	96.3	41.0	98.4	95.0	89.5	82.7	73.6	64.5	52.5	33.6	15.0	3.4	0.8	0.3
Meramec River	2.63	100.0	39.0	99.4	98.8	96.4	92.5	87.6	83.0	78.4	71.7	56.2	22.1	5.5	0.9
Long Island	2.66	97.5	41.0	99.5	98.2	95.5	89.0	80.3	71.8	61.5	47.6	32.5	15.2	6.6	2.4
Pelee Island	2.66	112.0	33.0	100.0	100.0	93.8	82.9	69.3	58.7	48.6	37.8	27.9	15.6	8.6	3.1
Plum Island	2.63	92.1	44.0	100.0	99.8	99.5	98.6	96.8	89.0	69.2	34.2	7.2	0.8	0.3	0.1
Cambridge	2.65	109.4	34.0	91.5	88.6	85.2	81.0	75.8	70.0	60.5	46.3	32.9	16.6	9.0	5.9
Sieve Opening, In	.....	.....	.....	0.263	0.185	0.131	0.093	0.065	0.046	0.0328	0.0232	0.0164	0.0116	0.0082	0.0058

TABLE 16.—PHYSICAL PROPERTIES OF MORTAR AND PLASTER SANDS

SAND	Specific Gravity	Weight per cu. ft. (Dry), Lb.	Computed Voids, Percent	Granular Analysis— Percent Passing Sieve No.							
				6	10	20	40	60	80	100	200
Coarse Lake	2.62	104.6	36.0	98.5	94.0	72.9	42.5	12.6	5.4	1.9	0.7
Fine Lake	2.62	100.3	39.0	99.5	98.3	94.2	76.1	42.6	16.4	10.4	0.6
Beach	2.63	91.7	44.0	99.0	97.0	90.7	75.5	48.8	31.3	17.5	0.3
Bank	2.64	99.6	39.0	99.9	99.9	99.7	98.5	89.7	47.5	20.9	.....
Sieve Opening, In	.....	.....	.....	0.131	0.065	0.0328	0.015	0.0087	0.0068	0.0058	0.0029

TABLE 17.—MORTAR STRENGTH TESTS OF CONCRETE SANDS

Proportion: 1 part Tidewater Portland cement, 3 parts sand (by weight)

Test specimens stored in water. Each result is the average of three tests.

SAND	Percent Water	Age, Days	Tensile Strength, Lb. per Sq. In.	Compressive Strength, Lb. per Sq. In.
Fox River	10.0	7	283	3390
Fox River	10.0	28	437	4980
Joliet	12.0	7	397	1347
Joliet	12.0	28	379	2920
Meramec River	13.5	7	171	921
Meramec River	13.5	28	346	1540
Long Island	13.0	7	176	954
Long Island	13.0	28	303	1910
Pelee Island	11.0	7	245	763
Pelee Island	11.0	28	412	3320
Plum Island	12.0	7	258	1600
Plum Island	12.0	28	269	1621

TABLE 18.—CHEMICAL ANALYSES OF COARSE AGGREGATES

AGGREGATE	Insoluble in Acid	Silica, Soluble	Al_2O_3 and R_2O_3	CaO	MgO	CO_2	SO_3	Loss* on Ignition
Chicago limestone	5.00	.....	0.44	29.16	20.40	45.35	.....	1.00
Joliet gravel	21.28	0.24	0.82	23.30	16.29	35.70	.....	1.16
Meramec R. gravel	96.06	0.50	2.36	Trace	Trace	.....	None	0.92
New York trap rock	86.08	0.24	8.22	1.05	1.93	.....	.....	.....
Rockport granite	97.31a	.....	.....	.....	.....	.....	.....	.....
Cleveland sandstone	96.59b	.....	.....	.....	.....	.....	.....	.....
Hard coal cinders	90.88	0.13	0.62	0.41	0.25	.....	0.03	6.97
Westfield trap rock	84.63	1.48	7.48	1.63	2.85	2.65	.....	0.19

* Other than CO_2.
(a) Soluble content too small to warrant further analysis. Tests indicate a normal granite.
(b) Normal sandstone.

TABLE 19.—PHYSICAL PROPERTIES OF COARSE AGGREGATES

AGGREGATE	Apparent Specific Gravity	Weight per Cu. Ft. (Dry), Lb.	Computed Voids, Percent	Percent Passing Sieves Size of Openings in In.						
				1½	1	¾	½	¼	0.131	0.093
Chicago limestone	2.61	86.50	46.9	100.0	98.5	76.5	33.0	6.0	0.14	0.11
Joliet gravel	2.61	101.25	37.9	100.0	95.7	83.5	56.5	6.5	.....	.....
Meramec R. gravel	2.45	95.0	38.0	100.0	98.0	91.5	62.0	19.0	6.60	3.60
New York trap rock	2.87	95.6	46.6	100.0	98.3	83.3	50.8	11.8	0.20	0.10
Rockport granite	2.61	87.25	46.3	100.0	100.0	100.0	86.5	29.0	7.30	1.60
Cleveland sandstone	2.15	78.50	41.6	98.7	83.7	45.0	20.0	9.0	7.30	7.00
Hard coal cinders	1.52	50.75	46.5	98.8	93.5	89.0	68.7	37.3	36.0	34.0
Westfield trap rock	2.94	90.0	51.0	100.0	100.0	100.0	100.0	58.3	38.8	19.4

TABLE 20.—MINERALOGICAL COMPOSITION OF CONCRETE AGGREGATES

AGGREGATE	Calcite, $CaCO_3$	Dolomite, $CaMg(CO_3)_2$	Clay, Hydrous Aluminum Silicates	Limonite (Iron Rust), $2Fe_2O_3.3H_2O$	Chert, $SiO_2.XH_2O$	Quartz, SiO_2	Feldspars, $KAlSi_3O_8$ $NaAlSi_3O_8$ $CaAl_2Si_2O_8$	Ferromagnesians, (Fe, Mg, Ca) Al_2SiO_6 (Fe, Mg, Ca) SiO_3	Magnetite, Fe_3O_4	Other Minerals. Amounts, Where not Given, Are Less Than 1 Percent
Fox River sand	6	38	(2 in calcite and dolomite)	0.5	7	32	11	4	0.5	white mica, garnet titanite
Joliet sand	5	42	(3 in calcite and dolomite)	trace		42	7	3	trace	garnet, cinders
Meramec River sand	1			trace	16	83				
Long Island sand	trace		1 (slate)			82	14	2	0.7	garnet, cinders
Pelee Island sand	27	5	4		4	47	9	3	0.5	garnet, muscovite corundum
Plum Island sand	trace					83	14	2	0.3	garnet, muscovite
Chicago limestone		94.5	5	0.5						
Joliet gravel		86	(5 in dolomite)			7	5	2		
Meramec River gravel	trace			trace	98	2				
New York trap rock						trace	71	26	2	apatite
Rockport granite			trace	trace		35	59	5	trace	zircon
Cleveland sandstone			1.5	0.5		98				
Hard coal cinders										85 slag and cinder 10 anthracite 5 ash

TABLE 21.—COMPRESSIVE STRENGTH AND MODULUS OF ELASTICITY OF CONCRETE IN COLUMNS AND COVERINGS

Cylinder No.	Kind of Concrete	Age, Days	Percent Water	Ultimate Stress, Lb. per Sq. In.	Modulus of Elasticity, Lb. per Sq. In.		
					Unit Stress		
					450	650	850
22–1	1:2:4 Chicago limestone	28	.....	912	.....	.....	.....
22–2	do	29	.....	1049	.....	.....	.....
22–3	do	407	.....	1888	2, 060, 000	2, 490, 000	2, 780, 000
22–4	do	407	.....	1780	2, 390, 000	2, 400, 000	2, 310, 000
28–1	do	28	.....	2935	.....	.....	.....
28–3	do	28	.....	3308	.....	.....	.....
28–2	do	438	.....	3830	.....	.....	5, 270, 000
28–4	do	438	.....	3470	4, 470, 000	4, 620, 000	4, 400, 000
28A–1	do	28	.....	1738	.....	.....	.....
28A–3	do	28	.....	1647	.....	.....	.....
28A–2	do	439	.....	2370	4, 300, 000	3, 960, 000	3, 550, 000
28A–4	do	439	.....	2213	.....	3, 900, 000	3, 500, 000
33–1	do	25	12.1	1489	.....	.....	.....
33–3	do	25	12.1	1728	.....	.....	.....
33–2	do	452	12.1	1956	2, 660, 000	2, 300, 000	1, 990, 000
33–4	do	452	12.1	2273	4, 000, 000	3, 730, 000	3, 350, 000
33A–1	do	29	12.9	1255	.....	.....	.....
33A–3	do	29	12.9	1676	.....	.....	.....
33A–2	do	456	12.9	1574	2, 950, 000	2, 680, 000	2, 450, 000
33A–4	do	456	12.9	1881	3, 350, 000	3, 100, 000	2, 950, 000
70–1	do	28	.....	2027	.....	.....	.....
70–3	do	28	.....	1778	.....	.....	.....
70–2	do	434	.....	3182	4, 280, 000	3, 950, 000	3, 860, 000
70–4	do	434	.....	2743	3, 750, 000	3, 730, 000	3, 490, 000
72–1	do	29	13.1	1010	.....	.....	.....
72–3	do	29	12.7	1582	.....	.....	.....
72–2	do	521	13.1	1656	2, 730, 000	2, 500, 000	2, 250, 000
72–4	do	521	12.7	2114	.....	.....	2, 970, 000
74–1	do	28	12.0	704	.....	.....	.....
74–3	do	28	11.8	739	.....	.....	.....
74–2	do	523	12.0	922	1, 420, 000	760, 000	250, 000
74–4	do	523	11.8	935	1, 760, 000	1, 160, 000	490, 000
101–5	do	28	12.7	1495	.....	.....	.....
101–6	do	520	12.7	2485	7, 970, 000	5, 990, 000	5, 080, 000
102–1	do	28	12.2	1455	.....	.....	.....
102–2	do	514	12.2	2363	4, 050, 000	3, 760, 000	3, 740, 000
111–1	do	28	12.5	1554	.....	.....	.....
111–2	do	520	12.5	2735	5, 000, 000	4, 320, 000	4, 450, 000
112–5	do	28	10.6	1493	.....	.....	.....
112–6	do	521	10.6	2042	3, 640, 000	3, 460, 000	3, 350, 000
19–1	1:3:5 Chicago limestone	29	.....	2056	.....	.....	.....
19–2	do	29	.....	2168	.....	.....	.....
19–3	do	414	.....	2561	4, 000, 000	3, 940, 000	3, 790, 000
19–4	do	414	.....	1880	3, 350, 000	3, 160, 000	3, 120, 000
35–1	do	32	12.8	1284	.....	.....	.....
35–3	do	32	12.8	1618	.....	.....	.....
35–2	do	505	12.8	1540	3, 270, 000	2, 940, 000	2, 520, 000
35–4	do	505	12.8	1468	4, 070, 000	3, 760, 000	3, 470, 000
41–1	do	29	13.0	1355	.....	.....	.....
41–3	do	29	13.4	1132	.....	.....	.....
41–2	do	452	13.0	1485	2, 830, 000	2, 520, 000	2, 300, 000
41–4	do	452	13.4	1313	2, 590, 000	2, 450, 000	2, 200, 000
42–1	do	28	13.3	1169	.....	.....	.....
42–3	do	27	12.6	1042	.....	.....	.....
42–2	do	450	13.3	1943	3, 450, 000	3, 180, 000	3, 150, 000
42–4	do	450	12.6	1216	2, 350, 000	2, 000, 000	1, 640, 000
55–1	do	29	13.7	718	.....	.....	.....
55–3	do	29	15.2	577	.....	.....	.....
55–2	do	483	13.7	794	1, 330, 000	490, 000	.....
55–4	do	483	15.2	678	1, 630, 000	.....	.....
56–1	do	28	13.3	811	.....	.....	.....
56–3	do	28	12.9	614	.....	.....	.....
56–2	do	492	13.3	682	1, 280, 000	400, 000	.....
56–4	do	492	12.9	869	1, 280, 000	750, 000	.....
57–1	do	28	12.7	912	.....	.....	.....
57–3	do	28	12.9	947	.....	.....	.....
57–2	do	483	12.7	921	1, 600, 000	900, 000	350, 000

TABLE 21.—COMPRESSIVE STRENGTH AND MODULUS OF ELASTICITY OF CONCRETE IN COLUMNS AND COVERINGS—Continued

Cylinder No.	Kind of Concrete	Age, Days	Per-cent Water	Ultimate Stress, Lb. per Sq. In.	Modulus of Elasticity, Lb. per Sq. In.		
					Unit Stress		
					450	650	850
57–4	1:3:5 Chicago limestone	483	12.9	1040	2,030,000	1,550.000	1,000,000
76–1	do	59	13.7	523	190,000	.....	.....
76–2	do	59	13.7	688	340,000	240,000	.....
77–1	do	59	13.7	842	1,360,000	950,000	.....
77–2	do	59	13.7	508	210,000	.....	.....
106–1	do	28	13.9	767	.....	.....	.....
106–4	do	28	13.9	813	.....	.....	.....
106–2	do	532	13.9	820	1,770,000	1,200,000	.....
106–3	do	532	13.9	946	2,230,000	1,460,000	1,060,000
16–1	1:2:4 New York trap	29	.....	1023	.....	.....	.....
16–2	do	29	.....	1163	.....	.....	.....
16–3	do	416	.....	2512	4,750,000	4,340,000	3,860,000
16–4	do	416	.....	1219	1,100,000	650,000	330,000
18–1	do	28	.....	1158	.....	.....	.....
18–2	do	28	.....	1764	.....	.....	.....
18–3	do	419	.....	1529	1,900,000	1,570,000	1,030,000
18–4	do	419	.....	1884	3,320,000	2,600,000	1,840,000
29–1	do	29	13.1	1494	.....	.....	.....
29–3	do	29	13.6	1239	.....	.....	.....
29–2	do	437	13.1	1807	2,570,000	2,390,000	2,210,000
29–4	do	437	13.6	1985	3,140,000	2,870,000	2,640,000
36–1	do	29	12.2	1490	.....	.....	.....
36–3	do	29	12.0	1548	.....	.....	.....
36–2	do	445	12.2	2284	2,750,000	2,740,000	2,600,000
36–4	do	445	12.0	2118	3,100,000	2,960,000	2,650,000
37–1	do	30	12.6	995	.....	.....	.....
37–3	do	30	12.7	1668	.....	.....	.....
37–2	do	504	12.6	2278	2,780,000	2,620,000	2,450,000
37–4	do	504	12.7	1103	820,000	530,000	350,000
40–1	do	28	11.2	1797	.....	.....	.....
40–3	do	28	11.1	1780	.....	.....	.....
40–2	do	502	11.2	2180	3,140,000	2,960,000	2,750,000
40–4	do	502	11.1	2300	4,170,000	3,450,000	3,150,000
46–1	do	28	13.9	1418	.....	.....	.....
46–3	do	29	13.9	1641	.....	.....	.....
46–2	do	455	13.9	1659	1,870,000	1,700,000	1,460,000
46–4	do	455	13.9	2082	2,730,000	2,510,000	2,300,000
60–1	do	27	14.6	948	.....	.....	.....
60–3	do	27	14.6	1028	.....	.....	.....
60–2	do	490	14.6	1490	2,510,000	2,140,000	1,810,000
60–4	do	490	13.8	1039	1,940,000	1,530,000	1,200,000
71–1	do	29	13.7	1572	.....	.....	.....
71–3	do	29	13.0	957	.....	.....	.....
71–2	do	451	13.7	1446	2,450,000	1,970,000	1,650,000
71–4	do	451	13.0	1109	1,000,000	400,000	230,000
73–1	do	28	.....	2239	.....	.....	.....
73–3	do	28	.....	2484	.....	.....	.....
73–2	do	443	.....	2658	4,180,000	3,960,000	3,720,000
73–4	do	443	.....	3224	3,900,000	4,060,000	3,830,000
75–1	do	28	10.0	1361	.....	.....	.....
75–3	do	28	10.3	1756	.....	.....	.....
75–2	do	462	10.0	1918	1,500,000	1,610,000	1,740,000
75–4	do	462	10.3	2318	2,790,000	2,770,000	2,700,000
101–3	do	28	13.2	1307	.....	.....	.....
101–4	do	541	13.2	2115	4,850,000	4,480,000	4,090,000
102–5	do	28	11.9	1510	.....	.....	.....
102–6	do	516	11.9	2692	3,020,000	3,060,000	3,070,000
104–1	do	32	11.1	978	.....	.....	.....
104–2	do	527	11.1	1542	3,900,000	3,510,000	2,350,000
20–1	1:3:5 New York trap	30	.....	1254	.....	.....	.....
20–2	do	30	.....	1158	.....	.....	.....
20–3	do	415	.....	1109	4,950,000	940,000	450,000
20–4	do	415	.....	958	8,700,000	330,000	.....
21–1	do	28	.....	702	.....	.....	.....
21–2	do	28	.....	678	.....	.....	.....
21–3	do	416	.....	874	590,000	270,000	.....
21–4	do	416	.....	986	900,000	520,000	250,000
54–1	do	29	12.6	780	.....	.....	.....
54–2	do	488	12.6	1012	1,330,000	1,850,000	530,000

TABLE 21.—COMPRESSIVE STRENGTH AND MODULUS OF ELASTICITY OF CONCRETE IN COLUMNS AND COVERINGS—Continued

Cylinder No.	Kind of Concrete	Age, Days	Percent Water	Ultimate Stress, Lb. per Sq. In.	Modulus of Elasticity, Lb. per Sq. In.		
					Unit Stress		
					450	650	850
15–1	1:2:4 Rockport granite	30	.....	1478	.....	.....	.....
15–2	do	30	.....	1181	.....	.....	.....
15–3	do	407	.....	1773	2,800,000	2,730,000	2,600,000
15–4	do	407	.....	2315	2,470,000	2,540,000	2,540,000
34–1	do	29	12.0	1638	.....	.....	.....
34–3	do	29	11.5	1308	.....	.....	.....
34–2	do	453	12.0	2211	2,800,000	2,570,000	2,390,000
34–4	do	453	11.5	1903	3,160,000	2,900,000	2,550,000
34A–1	do	28	12.7	853	.....	.....	.....
34A–3	do	28	10.7	1633	.....	.....	.....
34A–2	do	455	12.7	1157	2,060,000	1,750,000	1,350,000
34A–4	do	455	10.7	1833	2,680,000	2,670,000	2,560,000
103–1	do	28	13.0	993	.....	.....	.....
103–2	do	522	13.0	1645	3,620,000	3,100,000	2,960,000
113–1	do	32	12.2	1017	.....	.....	.....
113–2	do	527	12.2	1476	.....	2,480,000	2,220,000
50–1	1:3:5 Rockport granite	28	13.5	919	.....	.....	.....
50–3	do	28	13.5	621	.....	.....	.....
50–2	do	487	13.5	1082	2,150,000	1,800,000	1,350,000
50–4	do	487	13.5	1063	2,380,000	1,850,000	1,280,000
50A–1	do	28	11.5	734	.....	.....	.....
50A–3	do	28	11.5	660	.....	.....	.....
50A–2	do	509	11.5	1255	2,400,000	1,950,000	1,650,000
50A–4	do	509	11.5	1073	2,750,000	2,430,000	2,220,000
51–1	do	29	14.0	729	.....	.....	.....
51–4	do	29	14.0	703	.....	.....	.....
51–2	do	485	14.0	924	1,480,000	1,130,000	750,000
51–3	do	485	14.0	688	1,480,000	.....	.....
51A–1	do	29	14.1	683	.....	.....	.....
51A–4	do	29	14.1	822	.....	.....	.....
51A–2	do	507	14.1	877	2,170,000	1,260,000	640,000
51A–3	do	507	14.1	1272	2,270,000	2,230,000	2,000,000
31–1	1:2:4 Cleveland sandstone	30	14.0	2040	.....	.....	.....
31–3	do	30	14.2	1988	.....	.....	.....
31–2	do	501	14.0	2828	1,410,000	1,410,000	1,410,000
31–4	do	501	14.2	2531	1,230,000	1,230,000	1,200,000
43–1	do	29	14.0	746	.....	.....	.....
43–3	do	29	14.0	1855	.....	.....	.....
43–2	do	456	14.0	2128	1,420,000	1,290,000	1,230,000
43–4	do	456	14.0	2658	1,430,000	1,350,000	1,270,000
104–3	do	32	14.9	1535	.....	.....	.....
104–4	do	527	14.9	2720	2,270,000	2,020,000	1,670,000
44–1	1:3:5 Cleveland sandstone	29	14.7	1418	.....	.....	.....
44–3	do	29	14.7	661	.....	.....	.....
44–2	do	459	14.7	1662	4,130,000	3,040,000	2,050,000
44–4	do	459	14.7	1042	880,000	720,000	600,000
11–1	1:2:4 Joliet gravel	27	11.3	2365	.....	.....	.....
11–4	do	27	11.3	1764	.....	.....	.....
11–2	do	468	11.3	2190	3,030,000	3,100,000	3,120,000
11–3	do	468	11.3	2522	3,630,000	3,530,000	3,400,000
14–1	do	28	.....	1070	.....	.....	.....
14–2	do	28	.....	1192	.....	.....	.....
14–3	do	406	.....	2177	.....	3,670,000	3,470,000
14–4	do	406	.....	1268	3,380,000	2,940,000	2,620,000
30–1	do	28	.....	762	.....	.....	.....
30–3	do	28	.....	1908	.....	.....	.....
30–2	do	440	.....	898	2,250,000	1,550,000	620,000
30–4	do	440	.....	2690	3,240,000	3,210,000	3,200,000
38–1	do	29	10.8	1080	.....	.....	.....
38–3	do	29	10.8	1290	.....	.....	.....
38–2	do	451	10.8	1872	3,350,000	3,000,000	2,840,000
38–4	do	451	10.8	1721	3,500,000	3,220,000	3,130,000
101–1	do	28	11.0	1401	.....	.....	.....
101–2	do	520	11.0	2408	3,810,000	3,750,000	3,600,000
102–3	do	28	11.0	1720	.....	.....	.....
102–4	do	514	11.0	2951	4,880,000	4,370,000	4,130,000

TABLE 21.—COMPRESSIVE STRENGTH AND MODULUS OF ELASTICITY OF CONCRETE IN COLUMNS AND COVERINGS—Concluded

Cylinder No.	Kind of Concrete	Age, Days	Percent Water	Ultimate Stress, Lb. per Sq. In.	Modulus of Elasticity, Lb. per Sq. In.		
					Unit Stress		
					450	650	850
112-1	1:2:4 Joliet gravel	28	9.7	2004			
112-2	do	521	9.7	2973	4,290,000	4,160,000	4,140,000
39-1	1:2:4 Meramec R. gravel	28		2053			
39-3	do	28		1220			
39-2	do	437		2580	3,450,000	3,500,000	3,430,000
39-4	do	437		1721	3,550,000	3,500,000	3,430,000
45-1	do	25	11.8	2056			
45-3	do	25	12.3	1987			
45-2	do	447	11.8	2707	4,600,000	4,370,000	4,040,000
45-4	do	447	12.3	2374	3,870,000	3,920,000	3,900,000
112-3	do	28	10.7	1719			
112-4	do	521	10.7	2952	3,310,000	3,510,000	3,600,000
17-1	1:1½:4½ Hard coal cinders	28		671			
17-2	do	28		1307			
17-3	do	408		1691	1,730,000	1,500,000	1,330,000
17-4	do	408		2072	1,900,000	1,770,000	1,670,000
32-1	1:2:5 Hard coal cinders	29	22.9	495			
32-3	do	29	22.9	728			
32-2	do	504	22.9	970	1,080,000	900,000	700,000
32-4	do	504	22.9	977	1,250,000	1,000,000	610,000
32A-1	do	32	17.3	628			
32A-3	do	32	18.5	759			
32A-2	do	498	17.3	950	900,000	720,000	500,000
32A-4	do	498	18.5	796	750,000	530,000	
47-1	do	28	22.1	949			
47-3	do	28	22.4	907			
47-2	do	447	22.1	1335	1,070,000	1,050,000	970,000
47-4	do	447	22.4	1258	1,150,000	1,030,000	890,000
52-1	do	28	22.3	901			
52-2	do	28	22.3	914			
52-3	do	492	22.3	929	1,140,000	920,000	580,000
52-4	do	492	22.3	879	1,300,000	920,000	450,000
53-1	do	30	29.1	306			
53-3	do	30	29.1	492			
53-2	do	494	29.1	635	1,000,000		
53-4	do	494	29.1	691	1,190,000	670,000	
104-5	do	32	22.9	797			
104-6	do	527	22.9	1313	1,340,000	1,280,000	1,190,000
12-1	1:1½:3 Westfield trap	28		2397			
12-2	do	28		1312			
12-3	do	534		3880	3,520,000	3,370,000	3,270,000
12-4	do	533		4180	4,890,000	4,450,000	4,130,000

TABLE 22.—COMPRESSIVE STRENGTH AND MODULUS OF ELASTICITY OF CONCRETE IN HEAD PROTECTIONS

Cylinder No.	Kind of Concrete	Age, Days	Percent Water	Ultimate Stress, Lb. per Sq. In.	Modulus of Elasticity, Lb. per Sq. In.		
					Unit Stress		
					450	650	850
49–1	1:2:4 Chicago limestone	28	11 2	1473			
49–3	do	28	11. 2	1510			
49–2	do	489	11. 2	2673	8, 900, 000	6, 000, 000	6, 000, 000
49–4	do	489	11. 2	2178	3, 870, 000	3, 320, 000	3, 140, 000
59–1	do	29	10. 5	990			
59–3	do	29	10. 5	962			
59–2	do	494	10. 5	1085	2, 170, 000	1, 370, 000	830, 000
59–4	do	494	10. 5	1319	3, 780, 000	2, 870, 000	2, 250, 000
*64–1	do	27	9. 8	3428			
*64–4	do	27	9. 8	3119			
*64–2	do	502	9. 8	4036	8, 810, 000	7, 940, 000	7, 580, 000
*64–3	do	502	9. 8	4000	4, 230, 000	4, 060, 000	4, 060, 000
65–1	do	27	9. 7	2120			
65–4	do	27	9. 7	1951			
65–2	do	502	9. 7	3157	6, 500, 000	5, 990, 000	5, 830, 000
65–3	do	502	9. 7	2813	4, 300, 000	4, 050, 000	3, 770, 000
69–1	do	29	10. 9	1874			
69–3	do	29	10. 9	1698			
69–2	do	497	10. 9	2469	4, 150, 000	3, 880, 000	3, 750, 000
69–4	do	497	10. 9	2941	7, 080, 000	5, 620, 000	5, 000, 000
A–1	do	28	9. 6	2058			
A–3	do	28	9. 6	1917			
A–2	do	520	9. 6	2960	4, 040, 000	4, 020, 000	3, 810, 000
B–1	do	28	11. 2	1986			
B–3	do	28	11. 2	2197			
B–2	do	520	11. 2	3131	4, 060, 000	3, 940, 000	3, 770, 000
B–4	do	524	11. 2	2282	3, 400, 000	3, 420, 000	3, 300, 000
B–5	do	524	11. 2	2527	4, 100, 000	4, 070, 000	4, 090, 000
48–1	1:3:5 Chicago limestone	28	11. 5	961			
48–3	do	28	11. 5	1736			
48–2	do	489	11. 5	1358	2, 810, 000	2, 710, 000	2, 400, 000
48–4	do	489	11. 5	2397	3, 750, 000	3, 710, 000	3, 570, 000
62–1	do	28	11. 2	881			
62–3	do	28	11. 2	1324			
62–2	do	492	11. 2	1731	4, 580, 000	4, 260, 000	3, 730, 000
62–4	do	492	11. 2	1613	2, 900, 000	2, 770, 000	2, 370, 000
63–1	1:2:4 New York trap	27	11. 1	1403			
63–2	do	27	11. 1	1280			
63–3	do	496	11. 1	2175	2, 750, 000	2, 480, 000	2, 380, 000
63–4	do	496	11. 1	1872	3, 090, 000	2, 770, 000	2, 430, 000
*67A–1	do	28	10. 3	1945			
*67A–2	do	28	10. 3	1998			
*67A–3	do	493	10. 3	2573		7, 950, 000	6, 480, 000
*67A–4	do	493	10. 3	2406	4, 890, 000	4, 080, 000	1, 780, 000
49–5	1:2:4 Rockport granite	28	10. 6	1395			
49–7	do	28	10. 6	1509			
49–6	do	489	10. 6	2212	6, 650, 000	4, 230, 000	3, 640, 000
49–8	do	489	10. 6	2233	3, 790, 000	3, 490, 000	2, 900, 000
68–1	do	30	9. 8	1307			
68–4	do	30	9. 8	1495			
68–2	do	494	9. 8	1606	3, 580, 000	2, 950, 000	2, 570, 000
68–3	do	494	9. 8	1930	4, 080, 000	4, 000, 000	3, 500, 000
58–1	1:3:5 Rockport granite	28	13. 4	841			
58–4	do	28	13. 4	946			
58–2	do	498	13. 4	1172	2, 510, 000	2, 070, 000	1, 670, 000
58–3	do	498	13. 4	1263	2, 430, 000	2, 030, 000	1, 770, 000
105–5	1:2:4 Joliet gravel	28	10. 5	1128			
105–8	do	28	10. 5	1355			
105–6	do	538	10. 5	1921	3, 480, 000	3, 330, 000	3, 100, 000
105–7	do	538	10. 5	1723	2, 800, 000	2, 560, 000	2, 130, 000
106–5	do	29	10. 7	1396			
106–8	do	28	10. 7	1403			
106–6	do	532	10. 7	1653	3, 460, 000	3, 250, 000	3, 100, 000
106–7	do	532	10. 7	2076	3, 920, 000	3, 350, 000	3, 430, 000
105–1	1:2:4 Meramec R. gravel	28	9. 8	1721			
105–4	do	28	9. 8	1691			
105–2	do	538	9. 8	2699	5, 030, 000	4, 600, 000	4, 230, 000
105–3	do	538	9. 8	2706	3, 040, 000	3, 090, 000	3, 060, 000
109–5	do	28	8. 7	2152			
109–7	do	28	8. 7	2021			
109–6	do	509	8. 7	3232	8, 100, 000	5, 710, 000	5, 000, 000
109–8	do	509	8. 7	3095	5, 060, 000	4, 350, 000	4, 190, 000

*Time of mixing, 2 min. For all other cylinders time of mixing was 1 min.

Cylinder No. 64–2 failed on third application of maximum load, No. 64–3 on second application of maximum load.

TABLE 23.—TESTS OF LIME

Sample No.	Quicklime		Hydrated Lime	
	1	2	1	2
Chemical Analysis:				
Impurities (SiO_2 and R_2O_3)	1.73%	1.70%	1.44%	1.31%
CaO	88.98	85.53	69.59	69.59
MgO	1.26	0.89	1.37	1.37
Loss on Ignition	8.22	8.22	27.02	27.07
CO_2 Average	0.66		4.89	
Fineness:				
Residue on No. 20 Sieve, Sample Slacked and Washed Through	1.25		....	
Residue on No. 30 Sieve, Original Sample, Dry	....		4.4	
Residue on No. 200 Sieve, Sample Washed Through	....		3.97	
Soundness: Pats—				
5 Hr. in Steam	....		Unsound	

TABLE 24.—TESTS OF CALCINED GYPSUM

	Western	Eastern
Chemical Analysis:		
Impurities (SiO_2 and R_2O_3)	3.55%	2.10%
CaO	36.77	36.93
SO_3	52.22	51.37
Loss Below 60°C.	1.23	1.12
Loss Between 60°C. and Tyrrell Burner	6.67	8.35
Time of Set:		
Neat, 55 percent Water	More than 7 hr.; less than 16 hr.	24 hr.
*Tensile Strength: Lb. per Sq. In.		
Neat, 55 percent Water	200.0	Easily crumbles
	190.0	No test
	192.0	obtainable
1 Gypsum: 3 Ottawa Sand, 22 percent Water	97.0	175.0
	102.0	150.0
	85.0	150.0

*Mortars made up with distilled water and after setting 24 hr. in air, were dried to constant weight below 60°C. and then broken.

TABLE 25.—COMPRESSIVE STRENGTH OF PORTLAND CEMENT AND LIME PLASTER

Test specimens 2-in. cubes stored in air

Cube No.	Proportion, Parts by Loose Volume of Materials as Used	Percent Water	Age, Days	Compressive Strength, Lb. per Sq.In.
23-1	1 Portland cement			
	1/10 Hydrated lime	15.78	28	1080
	2½ Coarse lake sand			
23-2	do	15.78	28	1055
23-3	do	15.78	512	1623
23-4	do	18.23	29	1488
23-6	do	18.10	29	1795
23-7	do	18.10	511	2385
23-8	do	15.23	28	2715
23-9	do	15.23	509	3828
24-1	do	16.23	28	1839
24-2	do	16.23	501	2855
24-3	do	17.79	29	1350
24-4	do	17.79	498	2140
24-5	do	12.87	29	933
24-6	do	12.87	496	1669
25-1	do	16.20	28	2520
25-2	do	16.20	487	3956
25-3	do	18.50	29	1319
25-4	do	18.50	484	2480
26-1	do	18.35	29	1655
26-2	do	18.35	500	3035
26-3	do	15.82	28	2685
26-4	do	15.82	498	3744
27-1	do	18.00	29	1645
27-2	do	18.00	501	3191
27-3	do	16.50	29	976
27-4	do	16.50	499	1934
78-1	do	19.52	38	818
78-2	do	19.52	38	795
78-3	do	19.52	38	1083
78-4	do	19.52	38	640
78-5	do	19.52	38	610
78-6	do	19.52	38	790
78-7	do	21.15	34	991
78-8	do	21.15	34	583
78-9	do	21.15	34	1035
110-1	do	21.60	28	1455
110-2	do	21.60	28	1200
110-3	do	21.60	28	1225
110-4	do	21.60	514	1805
110-5	do	21.60	514	1595
110-6	do	21.60	514	1946
110-7	do	17.20	28	1913
110-8	do	17.20	28	1748
110-9	do	17.20	28	1716
110-10	do	17.20	512	2181
110-11	do	17.20	512	3408
110-12	do	17.20	512	2639
110-13	do	19.00	28	1789
110-14	do	19.00	28	1863
110-15	do	19.00	28	1831
110-16	do	19.00	507	1960
110-17	do	19.00	507	2990
110-18	do	19.00	507	2753
110-19	do	19.02	29	2084
110-20	do	19.02	29	2020
110-21	do	19.02	29	1710
110-22	do	19.02	500	2838
110-23	do	19.02	500	2758
110-24	do	19.02	500	3248
77-4	1 Slaked lime	18.36	60	158
	2½ Fine lake sand			
77-5	do	18.36	60	149

TABLE 26.—COMPRESSIVE STRENGTH OF CLAY TILE MORTAR

Test specimens 2-in. cubes stored in air

Cube No.	Proportion, Parts by Loose Volume of Materials as Used	Percent Water	Age, Days	Compressive Strength, Lb. per Sq. In.
	1 Portland cement			
48–1	1 Slaked lime	24.00	28	955
	4 Bank or beach sand			
48–2	do	24.00	28	873
48–3	do	24.00	499	1040
48–4	do	24.00	499	769
48–5	do	27.50	28	763
48–6	do	27.50	512	833
49–1	do	25.00	28	421
49–2	do	25.00	498	485
49–3	do	27.15	28	410
49–4	do	27.15	498	405
50–1	do	26.00	28	273
50–2	do	26.00	496	361
51–1	do	25.70	28	385
51–2	do	25.70	497	358
51–3	do	28.45	28	278
51–4	do	28.45	497	308
51A–1	do	24.45	29	303
51A–2	do	24.45	516	313
53–1	do	24.40	28	445
53–2	do	24.40	503	370
56–1	do	24.60	28	455
56–2	do	24.60	28	399
56–3	do	24.60	500	565
56–4	do	24.60	500	550
57–1	do	22.70	29	513
57–2	do	22.70	497	700
58–1	do	24.80	28	288
58–2	do	24.80	504	373
58–3	do	24.10	28	340
58–4	do	24.10	504	348
58–5	do	28.00	28	283
58–6	do	28.00	504	325
59–1	do	25.90	28	395
59–2	do	25.90	498	333
59–3	do	24.75	28	368
59–4	do	24.75	497	270
60–1	do	25.00	28	296
60–2	do	25.00	505	285
60–3	do	26.60	28	360
60–4	do	26.60	505	358
61–1	do	28.05	29	513
61–2	do	28.05	29	390
61–3	do	28.05	524	473
61–4	do	28.05	524	628
61–5	do	23.50	29	415
61–6	do	23.50	524	480
62–1	do	25.30	28	475
62–2	do	25.30	483	551
63–1	do	28.25	28	318
63–2	do	28.25	505	350
68–1	do	29.50	28	310
68–2	do	29.50	503	270
68–3	do	24.10	28	285
68–4	do	24.10	503	323
69–1	do	21.80	516	463
69–3	do	24.45	29	318
69–4	do	24.45	516	338
69–5	do	21.45	28	853
69–6	do	21.45	515	940
69–7	do	22.65	28	653
69–8	do	22.65	28	663
69–9	do	22.65	493	773

TABLE 26.—COMPRESSIVE STRENGTH OF CLAY TILE MORTAR—Concluded

Test specimens 2-in. cubes stored in air

Cube No.	Proportion, Parts by Loose Volume of Materials as Used	Percent Water	Age, Days	Compressive Strength, Lb. per Sq. In.
69–10	1 Portland cement 1 Slaked lime 4 Bank or beach sand	22.65	493	708
76–1	do	23.75	60	495
76–2	do	23.75	60	571
76–3	do	23.75	60	588
77–1	do	23.75	60	380
77–2	do	23.75	60	385
77–3	do	23.75	60	360
52–2	1 Portland cement 2 Slaked lime 6 Bank sand	29.40	38	228
52–5	do	30.30	29	100
54–5	do	26.60	28	159
54–6	do	26.60	498	216
54–1	1 Portland cement ½ Hydrated lime 6 Bank sand	20.90	28	303
54–3	do	20.90	28	395
54–4	do	20.90	499	626

TABLE 29.—COMPRESSIVE STRENGTH OF GYPSUM FILLING

Test specimens 8 by 16 in. cylinders

Cylinder No.	MATERIAL	Proportions, Parts by Loose Volume of Materials as Used	*Percent Water	Age, Days	Ultimate Strength, Lb. per Sq. In.
66–1	Eastern gypsum filling	1 Eastern calcined gypsum 1 Fine lake sand 4 Broken gypsum blocks	54.70	31	15
66–5	do	do	61.20	29	31
66–6	do	do	61.20	483	95
67A–5	do	do	60.15	29	22
67A–6	do	do	60.15	29	17
67A–7	do	do	60.15	489	105
67A–8	do	do	60.15	489	85
109–3	Specimen collapsed on removal	of mould 12 days after placement	52.20	.....	.....
109–4	Broken during storage	.....	52.20	.....	.....
108–1	Western gypsum filling	1 Western calcined gypsum 1 Fine lake sand 4 Broken gypsum blocks	63.30	28	27
108–2	do	do	63.30	28	33
108–3	do	do	63.30	513	42
108–4	do	do	63.30	513	32
109–1	do	do	63.10	28	29
109–2	do	do	63.10	509	40

* Based on total weight of dry materials in mixture.

TABLE 27.—COMPRESSIVE STRENGTH OF GYPSUM MORTAR AND PLASTER

Test specimens 2-in. cubes stored in air

Cube No.	Proportion, Parts by Loose Volume of Materials as Used	*Percent Water	Age, Days	Compressive Strength, Lb. per Sq. In.
64–1	1 Western calcined gypsum			
	3 Fine lake sand	23.40	30	220
64–3	do	23.40	30	145
64–4	do	23.40	523	118
64–5	do	23.40	523	94
65–1	do	22.20	29	233
65–2	do	22.20	29	275
65–3	do	22.20	29	275
65–4	do	22.20	522	202
65–5	do	22.20	522	214
66–1	1 Eastern calcined gypsum			
	3 Fine lake sand	23.50	29	43
66–2	do	23.50	29	13
66–3	do	23.50	29	15
66–5	do	23.50	504	35
67–1	do	23.10	28	193
67–2	do	23.10	28	105
67–3	do	23.10	28	213
67–5	do	23.10	520	152
67–6	do	23.10	503	160
67A–1	do	22.85	31	215
66A–2	do	22.85	31	130
67A–3	do	22.85	31	188
67A–6	do	22.85	497	83
108–1	do	22.60	27	189
108–2	do	22.60	27	201
108–3	do	22.60	489	83
108–4	do	20.60	27	240
108–5	do	20.60	489	139
108–6	do	20.60	489	172
109–1	do	22.20	28	70
109–2	do	22.20	28	86
109–3	do	22.20	515	75
109–4	do	22.60	515	139
109–5	do	22.60	28	200
109–6	do	22.60	515	222
76–4	1 Fibered gypsum plaster			
	3 Fine lake sand	17.20	60	443
76–5	do	17.20	60	468
76–6	1 Calcined gypsum			
	2 Hydrated lime	61.00	60	958
77–6	do	61.00	60	1029

*Based on total weight of dry materials in mixture.

TABLE 28.—STRENGTH TESTS OF MORTAR AND PLASTER

Test specimens, 1-in. briquettes for tension tests and 2-in. cubes for compression tests made in laboratory. Each value is the average of three tests.

Mortar or Plaster	Proportion, Parts by Loose Volume of Materials as Used	*Percent Water	Method of Storage	Age, Days	Strength, Lb. per Sq. In.	
					Tensile	Compressive
Clay tile mortar	1 Portland cement 1 Slaked lime 4 Bank sand	25.0	1 day in damp closet, remainder of period in water	7	67	257
do	do with beach sand	25.0	do	7	72	236
do	do with Ottawa sand	25.0	do	7	65	174
do	do with bank sand	25.0	do	28	97	425
do	do with beach sand	25.0	do	28	114	496
do	do with Ottawa sand	25.0	do	28	91	369
do	do with Ottawa sand	.....	do	28	161	848
do	do with bank sand	25.0	1 day in damp closet, remainder of period in air	28	.....	328
do	do with Ottawa sand	25.0	do	28	.....	190
do	do with bank sand	25.0	do	365	.....	442
do	do with Ottawa sand	25.0	do	365	.....	259
Portland cement plaster	1 Portland cement 1/10 Hydrated lime 2½ Coarse lake sand	16.7	1 day in damp closet, remainder of period in water	7	144	418
do	do with Ottawa sand	16.7	do	7	114	363
do	do with coarse lake sand	16.7	do	28	197	1105
do	do with Ottawa sand	16.7	do	28	154	721
do	do with coarse lake sand	16.7	1 day in damp closet, remainder of period in air	28	.....	1010
do	do with Ottawa sand	16.7	do	28	.....	616
do	do with coarse lake sand	16.7	do	365	.....	1297
do	do with Ottawa sand	16.7	do	365	.....	963
Western gypsum block mortar	1 Western gypsum 3 Fine lake sand	22.0	1 day in air, then dried to constant weight below 60° C.	2	126	.....
do	do with Ottawa sand	22.0	do	2	95	.....
Eastern gypsum block mortar	1 Eastern gypsum 3 Fine lake sand	22.5	do	2	202	.....
do	do with Ottawa sand	22.5	do	2	158	.....
Western gypsum block mortar	1 Western gypsum 3 Fine lake sand	22.0	In air for whole period.	28	.....	491
do	do with Ottawa sand	22.0	do	28	.....	343
do	do with fine lake sand	22.0	do	365	.....	647
do	do with Ottawa sand	22.0	do	365	.....	517
Eastern gypsum block mortar	1 Eastern gypsum 3 Fine lake sand	22.5	do	28	.....	786
do	do with Ottawa sand	22.5	do	28	.....	415
do	do with fine lake sand	22.5	do	365	.....	1180
do	do with Ottawa sand	22.5	do	365	.....	698

*Based on total weight of dry materials in mixture.

TABLE 30.—CLASSIFICATION AND DESCRIPTION OF HOLLOW CLAY TILE

Tile No.	Nominal Thickness, In.	Type of Clay	District	Hardness	Color	Average Outside Dimensions, In.			Thickness of Side Shells, In.	Cross Webs		Scoring		
						Thickness	Width	Length		Number	Av. Thickness, In.	Number	Shape	Dimensions, In.
A	2	Surface clay	Chicago, Ill.	Soft	Light red	2.06	12.00	12.00	0.80	4	0.80	22	Rectangular	0.12 by 0.50
A	4	do	do	Soft	Light red	4.09	12.20	12.00	0.79	4	0.71	22	do	0.12 by 0.50
B	2	do	Boston, Mass.	Medium	Red	2.10	6.10	12.00	0.65	3	0.68	12	Half round	0.30 diam.
B	4	do	do	Medium	Red	4.03	12.30	11.70	0.83	5	0.75	24	do	0.40 diam.
C	2	Semi-fire clay	Waynesburg, O.	Hard	Buff	2.07	12.02	11.90	0.55	4	0.50	24	Rectangular	0.12 by 0.45
C	4	do	do	Hard	Buff	3.98	11.84	11.85	0.53	4	0.46	28	do	0.12 by 0.45
D	2	Shale	Cleveland, O.	Hard	Red	2.01	11.90	11.65	0.52	4	0.50	26	do	0.02 by 0.25
D	4	do	do	Hard	Red	3.96	11.62	11.85	0.60	4	0.56	40	do	0.10 by 0.30
E	2	Semi-fire clay	Perth Amboy, N. J	Medium	Buff	2.00	7.96	12.20	0.70	4	0.82	22	Half round	0.20 diam.
E	4	do	do	Medium	Buff	3.85	11.87	11.87	0.67	4	0.64	38	do	0.30 diam.
F	2-in. curved	do	do	Soft	Light red	2.09	10.56	12.00	0.85	4	0.96	11	do	0.30 diam.
G	2-in. curved	do	do	Soft	Light red	2.02	14.00	12.00	0.83	4	1.45	16	do	0.35 diam.
H	2-in. curved	do	do	Soft	Light red	2.55	10.94	12.00	0.76	4	0.67	11	Rectangular	0.10 by 0.37
												3	do	0.40 by 2.74

TABLE 31.—COMPRESSIVE STRENGTH OF HOLLOW CLAY TILE

Tile units are 12 in. wide and 12 in. long, except as noted

Sample No.	Kind of Title		Porosity, Percent of Volume	Absorption, Percent of Dry Weight	How Tested	Area under Load, Sq. In.	Maximum Load	
	Nominal Thickness, In.	Clay					Total, Lb.	Lb. per Sq. In.
A-1	2	Surface clay,	40.3	29.0	On end	19.6	44300	2260
A-2	2	Chicago	44.4	28.5	do	19.6	46860	2390
A-3	2	district	43.1	27.3	do	19.6	44650	2280
Average			42.6	28.3			45277	2310
A-4	2	do	44.5	28.5	On edge	16.2	18920	1170
A-5	2	do	43.7	27.7	do	16.2	32110	1980
A-6	2	do	41.9	26.2	do	16.2	22340	1380
Average			43.4	27.5			24456	1510
A-16	4	do			On end	25.2	53000	2100
A-19	4	do	34.4	27.6	do	25.2	57280	2270
A-20	4	do			do	25.2	63400	2520
Average							57913	2296
A-11	4	do	44.2	28.1	On edge	16.0	29320	1830
A-12	4	do	42.9	29.5	do	16.0	20250	1270
A-14	4	do	41.8	26.0	do	16.0	33190	2070
Average			42.9	27.5			27588	1723
*B-6	2	Surface clay,	28.8	15.0	On end	9.1	52830	5800
*B-8	2	Boston	27.3	15.0	do	9.1	48400	5310
*B-10	2	district	27.9	14.2	do	9.1	62450	6800
Average			28.0	14.7			54560	5990
*B-1	2	do	30.0	16.0	On edge	11.7	52450	4480
*B-2	2	do	28.6	15.0	do	11.7	58150	4970
*B-4	2	do	31.0	16.6	do	11.7	58040	4960
Average			29.8	15.8			56213	4796
B-12	4	do	30.8	16.4	On end	28.4	134350	4740
B-14	4	do	32.4	17.6	do	28.4	111400	3920
B-19	4	do	25.6	12.8	do	27.2	160930	5910
Average			29.6	15.6			135560	4856
B-11	4	do	28.8	15.0	On edge	15.3	68170	4460
B-13	4	do	28.0	14.6	do	15.3	69760	4560
B-17	4	do	29.8	15.6	do	14.1	56710	4020
Average			28.8	15.1			64880	4346
C-1	2	Ohio semi-fire clay	15.5	7.3	On end	14.1	98350	6980
C-5	2	do	15.4	7.0	do	14.1	123500	8770
C-6	2	do	22.5	11.2	do	14.7	94280	6410
Average			17.8	8.5			105376	7386
C-2	2	do	18.3	8.8	On edge	11.2	68870	6150
C-4	2	do	18.5	8.8	do	11.2	47040	4200
C-8	2	do	16.9	7.9	do	11.2	46920	4190
Average			17.9	8.8			54276	4846
C-13	4	do	18.5	9.0	On end	16.7	123610	7400
C-14	4	do	17.6	8.6	do	17.2	104850	6090
C-19	4	do	18.5	8.9	do	16.7	96050	5760
Average			18.2	8.8			108170	6416
C-15	4	do	16.0	8.4	On edge	9.6	58180	6060
C-18	4	do			do	10.0	36780	3680
Average							47480	4870

*Nominal width of tile, 6 in.

TABLE 31.—COMPRESSIVE STRENGTH OF HOLLOW CLAY TILE—Concluded

Tile units are 12 in. wide and 12 in. long, except as noted

Sample No.	Kind of Tile		Porosity, Percent of Volume	Absorption, Percent of Dry Weight	How Tested	Area under Load, Sq. In.	Maximum Load	
	Nominal Thickness, In.	Clay					Total, Lb.	Lb. per Sq. In.
D-1	2	Ohio shale	8.1	3.5	On end	14.3	114350	8000
D-3	2	do	22.1	10.8	do	14.1	112600	7990
D-9	2	do	14.3	7.2	do	14.5	109680	7560
Average			14.8	7.1			112210	7850
D-2	2	do	16.2	7.4	On edge	11.1	43300	3900
D-5	2	do	12.2	5.5	do	10.7	61980	5800
D-8	2	do	8.2	3.6	do	11.1	81450	7340
Average			12.2	5.5			62243	5680
D-11	4	do	12.0	5.2	On end	19.9	193550	9720
D-16	4	do	11.3	4.9	do	19.1	151770	7950
D-17	4	do	12.2	5.5	do	19.9	177940	8930
Average			11.8	5.2			174420	8866
D-13	4	do	16.3	7.4	On edge	11.7	95840	8190
D-14	4	do	13.8	6.5	do	11.7	61130	5220
D-19	4	do	9.6	3.8	do	11.5	117790	10230
Average			13.2	5.9			91686	7880
*E-5	2	Semi-fire clay, New Jersey district	28.3	17.2	On end	12.1	53240	4400
*E-6	2		30.4	17.1	do	12.1	51040	4210
*E-8	2				do	12.1	55620	4600
Average							53313	4402
*E-1	2	do	33.2	19.8	On edge	15.6	33670	2160
*E-2	2	do	33.5	19.6	do	15.9	29000	1820
*E-10	2	do	35.8	22.0	do	15.9	23100	1450
Average			34.1	20.4			28590	1810
E-15	4	do	25.3	14.2	On end	21.6	95650	4440
E-17	4	do	33.6	20.6	do	21.6	74660	3460
E-19	4	do	29.5	17.2	do	21.6	86250	4000
Average			29.4	17.3			85520	3966
E-12	4	do	29.0	16.2	On edge	12.6	39900	3170
E-13	4	do	33.2	19.8	do	12.6	24920	1980
E-14	4	do	33.9	20.4	do	12.6	20050	1590
Average			32.0	15.4			28290	2246
†F-5	2	Porous semi-fire clay, New Jersey district	37.6	23.2	On end	18.9	83270	4410
†F-8	2		41.8	27.0	do	19.9	54200	2720
†F-10	2		45.3	30.4	do	18.3	72630	3970
Average			41.5	23.8			70033	3700
†G-2	2	do	49.1	27.6	do	24.6	38060	1550
†G-3	2	do	50.2	39.1	do	23.9	33370	1400
†G-4	2	do	40.2	26.4	do	23.9	53440	2230
Average			46.5	31.0			41623	1726
†H-1	2½	do	51.2	39.0	do	19.9	27150	1360
†H-2	2½	do	46.9	34.3	do	18.0	32110	1780
†H-4	2½	do	41.4	27.7	do	20.8	40300	1940
Average			46.5	33.3			33086	1693

* Nominal width of tile, 8 in. † Curved tile.

TABLE 32.—TRANSVERSE STRENGTH OF HOLLOW TILE

Tile tested flatwise on side with center load and supports 10 in. apart. Tile units are 12 in. wide, except as noted.

Sample No.	Kind of Tile		Maximum Load, Lb.	Calculated Maximum Stresses, Lb. per Sq. In.	
	Nominal Thickness, In.	Clay		Bending, Outer Fiber	Shear, Center of Webs
A-7	2	Surface clay, Chicago district	1120	395	158
A-8	2	do	1050	370	148
A-9	2	do	830	292	117
A-10	2	do	1100	387	155
Average			1025	361	145
A-13	4	do	2600	254	146
A-15	4	do	1500	147	84
A-17	4	do	2500	244	140
A-18	4	do	1890	185	106
Average			2122	207	119
*B-3	2	Surface clay, Boston district	1000	625	202
*B-5	2	do	1520	950	307
*B-7	2	do	1280	800	258
*B-9	2	do	1240	775	250
Average			1260	787	254
B-15	4	do	5600	547	253
B-16	4	do	6900	675	312
B-18	4	do	5100	499	231
B-20	4	do	5250	513	237
Average			5712	558	283
C-7	2	Ohio semi-fire clay	1220	477	236
C-10	2	do	2040	798	394
Average			1630	637	315
C-20	4	do	2120	321	223
D-4	2	Ohio shale	2530	945	415
D-6	2	do	1630	608	267
D-7	2	do	3200	1194	525
Average			2452	916	402
D-12	4	do	4530	564	320
D-15	4	do	6250	778	441
D-18	4	do	4910	611	346
D-20	4	do	3280	408	231
Average			4742	590	335
†E-3	2	New Jersey semi-fire clay	1410	705	155
†E-4	2	do	1100	550	121
Average			1255	627	138
E-11	4	do	2010	272	142
E-16	4	do	1900	257	134
E-18	4	do	1330	180	94
E-20	4	do	2570	347	182
Average			1952	264	140

*Nominal width, 6 in. †Nominal width, 8 in.

TABLE 33.—TEMPERATURES OF VITRIFICATION AND FUSION OF CLAY TILE AND BRICK

No.	CLAY	Temperature of Vitrification, Deg. C.	Temperature Producing Softening, Deg. C.	Temperature of Fusion, Deg. C.
A	Surface clay, Chicago district	1120	1200	1240
B	Surface clay, Boston district	1120	1160	1180
C	Ohio semi-fire clay	1180	1450	1460
D	Ohio shale	1120	1380	1400
E	New Jersey semi-fire clay partition tile	1200	1480	1500
F	New Jersey porous semi-fire clay, 2 by 8-in. curved tile	....	1450	1470
G	New Jersey porous semi-fire clay, 2 by 12-in. curved tile	....	1510	1530
H	New Jersey porous semi-fire clay, 2½ by 11-in. curved tile	....	1450	1470
J	Chicago common brick	....	....	1145

TABLE 34.—POROSITY AND ABSORPTION OF CHICAGO COMMON BRICK

Brick No.	Specimen No.	Porosity, Percent of Volume	Absorption, Percent of Dry Weight
1	1	35.4	20.2
1	2	31.9	17.8
1	3	32.5	18.0
1	4	31.5	17.6
1	5	24.5	13.1
1	6	36.3	20.9
1	7	26.5	15.4
1	8	24.3	12.9
1	Average	30.4	16.9
2	1	43.1	26.8
2	2	42.1	26.9
2	3	40.1	24.4
2	4	39.9	24.4
2	5	40.3	24.7
2	6	40.9	25.1
2	7	40.3	24.6
2	8	40.6	24.9
2	Average	40.9	25.1

TABLE 35.—COMPRESSIVE STRENGTH OF CHICAGO COMMON BRICK

Sample No.	How Tested	Dimensions, In.			Area Under Load, Sq. In.	Maximum Load	
		Width	Thickness	Length		Total, Lb.	Lb. per Sq. In.
J-1	On end	3.45	2.20	....	7.59	32900	4330
J-2	do	3.75	2.25	....	8.44	14900	1760
J-4	do	3.50	2.25	....	7.88	27550	3500
Average							3200
J-3 ½ brick	On edge	....	2.25	2.9	6.52	7550	1160
J-5 do	do	....	2.15	3.35	7.20	26000	3610
J-7 do	do	....	2.35	5.40	12.69	15680	1240
J-8 do	do	....	2.35	4.10	9.64	16800	1740
J-9 do	do	....	2.25	3.40	7.67	15150	1980
J-10 do	do	....	2.25	4.55	10.24	20700	2020
Average							1960
J-3 ½ brick	On side	3.70	....	4.10	15.17	46010	3030
J-5 do	do	3.50	....	4.60	16.10	55800	3460
J-6 do	do	3.60	....	7.80	28.08	57730	2050
J-7 do	do	3.50	....	2.60	9.10	16490	1810
J-9 do	do	3.60	....	4.60	16.56	67910	4100
J-10 do	do	3.65	....	3.70	13.50	32900	2440
Average							2815

TABLE 36.—TRANSVERSE STRENGTH OF CHICAGO COMMON BRICK

Brick tested on side with center load and supports 7 in. apart

Sample No.	Dimensions, In.		Maximum Load, Lb.	Modulus of Rupture Lb. per Sq. In.
	Width	Thickness		
J-3	3.70	2.25	940	530
J-5	3.50	2.15	2700	1750
J-8	3.70	2.35	460	240
J-9	3.60	2.25	1540	890
J-10	3.65	2.25	1580	900
Average				862

TABLE 37.—POROSITY OF GYPSUM BLOCK

Sample No.	Kind of Block		Porosity, Percent of Total Volume
	Thickness	Gypsum	
K–4	4-in. hollow	Western	63.7
K–5	4-in. hollow	do	63.7
L–4	4-in. solid	do	63.7
L–5	4-in. solid	do	63.9
M–4	2-in. solid	do	63.6
M–5	2-in. solid	do	63.6
Average			63.7
N–4	4-in. solid	Eastern	58.6
N–5	4-in. solid	do	58.4
O–4	2-in. solid	do	62.6
O–5	2-in. solid	do	62.2
Average			60.4

TABLE 38.—COMPRESSIVE STRENGTH OF SOLID GYPSUM BLOCK

All blocks tested on edge

Sample No.	Gypsum	Dimensions, In.			Area Under Load, Sq. In.	Maximum Load	
		Thickness	Length	Width		Total, Lb.	Lb. per Sq. In.
L–1	Western	4	30.1	12	120.5	61100	507
L–2	do	4	30.1	12	120.5	44300	367
L–3	do	4	30.0	12	120.0	49080	409
Average							428
M–1	do	2	29.9	12	59.8	38250	640
M–2	do	2	29.9	12	59.8	23760	396
M–3	do	2	30.0	12	60.0	23840	398
Average							478
N–1	Eastern	4	25.7	16.5	103.0	67540	656
N–2	do	4	25.7	16.5	103.0	51340	498
N–3	do	4	26.2	16.5	105.0	52860	504
Average							553
O–1	do	2	26.1	16.5	52.2	23950	458
O–2	do	2	26.1	16.5	52.2	19205	365
O–3	do	2	26.1	16.5	52.2	21685	415
Average							413

TABLE 39.—TRANSVERSE STRENGTH OF SOLID GYPSUM BLOCK

All blocks tested flatwise on side with center load

Sample No.	Gypsum	Dimensions, In.			Span, In.	Weight of Block, Lb.	Maximum Load, Lb.	Modulus of Rupture, Lb. per Sq.In.
		Thickness	Length	Width				
L-7	Western	4	30.2	12	24	42.9	689	132
L-8	do	4	30.2	12	24	47.4	652	125
L-9	do	4	30.2	12	24	46.2	562	109
Average								122
M-7	do	2	29.8	12	24	22.7	266	206
M-8	do	2	29.9	12	24	22.1	260	201
M-9	do	2	29.9	12	24	22.8	294	227
Average								211
N-7	Eastern	4	26.1	16.5	20	58.9	1520	175
N-8	do	4	26.1	16.6	20	56.2	1438	165
N-9	do	4	26.0	16.6	20	58.2	1503	169
Average								170
O-7	do	2	25.9	16.4	20	24.4	305	143
O-8	do	2	26.2	16.3	20	27.7	245	117
O-9	do	2	25.9	16.4	20	25.7	311	146
Average								135

TABLE 40.—TRANSVERSE STRENGTH OF GYPSUM WALL BOARD

Samples, 18 in. long, were tested with center load and supports 16 in. apart

No.	Width, In.	Thickness, In.	Weight, Lb.	Condition of Board	Direction of Loading Bar	Maximum Load, Lb.
A-1	11.9	0.38	3.1	Dry	Parallel with grain of paper	37.0
A-2	12.0	0.39	3.1	Dry	do	39.0
A-3	12.0	0.39	3.1	Dry	do	37.5
Average						37.8
B-1	12.0	0.39	3.1	Dry after having been saturated	Parallel with grain of paper	39.0
B-2	11.9	0.38	3.0		do	35.5
Average						37.2
C-1	11.9	0.40	3.8	Saturated	Parallel with grain of paper	10.0
C-2	12.0	0.38	3.7	Saturated	do	6.0
Average						8.0
D-1	12.0	0.38	3.1	Dry	Perpendicular to grain of paper	90.5
D-2	12.0	0.34	3.2	Dry	do	96.0
D-3	12.0	0.39	3.2	Dry	do	92.0
Average						92.8
E-1	12.0	0.38	3.0	Dry after having been saturated	Perpendicular to grain of paper	89.5
E-2	12.0	0.38	3.1			87.0
Average						88.2
F-1	12.0	0.38	3.7	Saturated	Perpendicular to grain of paper	15.5

APPENDIX E.—PREVIOUS INVESTIGATIONS

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The first experimental investigations on the fire resistance of building columns were made abroad, principally in Germany. When iron came into use as a structural material in that country, it was thought that the possibility of constructing truly fireproof buildings was realized, for iron is not combustible. Extensive fires showed, however, that, while the structural framework did not burn, the building collapsed suddenly and without warning. It was early recognized that some wooden structures offered greater resistance to fire than those built of unprotected iron, a result that occasioned no little comment when first observed. In fact, iron as a structural material was for a time in considerable disrepute. At one time the use of unprotected cast-iron columns under main bearing walls was forbidden in Berlin but wrought-iron columns permitted. Subsequent to large fires in Berlin and Hamburg the reverse was true. Similar changes in opinion were evident as to whether cast and wrought iron columns should be given fire-protective coverings and whether such coverings should be removable or permanent.

1. BAUSCHINGER'S TESTS[3]

In 1884–1886 Prof. J. Bauschinger, of Munich, Germany, made two series of fire and water tests on building columns that were loaded in a horizontal testing machine and heated by wood fire in a wrought-iron trough placed under them. Water was applied to the top surface. Temperatures were measured by alloys of tin, lead, and silver, having computed melting points of 300, 400, 500, and 600° C, that were attached to rods and held against the surface of the column at the middle of the sides to obtain the average temperature. Deflections were measured in the vertical and horizontal directions by indicators attached to wires that were fastened to the column at the middle of its length. The columns were loaded to what were considered safe working loads and subjected to three successive fire and water tests, the surface temperatures it was aimed to attain when water was applied being generally 300° C, 400 to 600° C, and red heat above 600° C. These tests were made with the column ends fixed or restrained. If failure did not occur, a final test at red heat with the column ends unrestrained was made in most cases. The method of testing was largely determined by the fact that opinions differed as to whether damage to cast-iron columns in building fires was caused by the fire or by the application of water to the red-hot metal.

During the first part of each fire test the column deflected downward toward the fire, due to the unequal heating, but with increase of temperature greater uniformity obtained and the column straightened somewhat. The application of water to the upper surface caused another sharp deflection downward, which became less as the

[3] Mittheilungen aus den Mech. Tech. Lab. d. k. Tech. Hochschule, München, Heft 12, 1885; Heft 15, 1887.

column cooled on all sides, the final deflection being in some cases upward. The test conditions and effects approximate to some extent those pertaining to unprotected columns under exterior walls.

(*a*) First Series.—In the first series tests were made on six cast-iron, three wrought-iron, and 15 columns of other building materials, including Portland-cement mortar, brick, and several kinds of building stone.

The cast-iron columns had been rejected for building purposes because of uneven wall thickness, "cold shuts," and other defects, and were more or less ornamental in form, the types varying from the plain cylindrical and slightly tapering shafts to the form having an ornamental base for about one-third of the length and a tapering shaft, which in one case was deeply fluted. They had plain or highly ornamental capitals and were from 11 feet to 13.8 feet in length and from 5.8 to 7.6 inches in outside diameter, as measured at the midheight of the column, with average wall thicknesses of 0.40 to 1 inch, the thickness varying considerably within each column. Working loads of 3400 to 5800 pounds per square inch were applied, the stress depending on the slenderness ratio of the column.

As tested with restrained ends the cast-iron columns supported their full load in the fire and water tests, although cracks developed in some cases at the higher temperatures when water was applied. Maximum vertical deflections of 3½ inches were observed after application of water on columns of nearly uniform wall thickness. With columns of uneven wall thickness the deflections were larger. In the tests with unrestrained ends, on application of water the resulting deflection made the columns unable to support full load and caused some of them to break.

The wrought-iron columns were about 13 feet (4 meters) in length and consisted of one welded tube 5.04 inches outside diameter and 0.24 inch thick, and two built-up box columns, one of two 7-inch channels and two plates fastened together by bolts spaced about 16 inches on centers, and the other of two 7-inch I beams and two plates fastened with bolts spaced about 7 inches apart. The loads were 6200, 7600, and 6700 pounds per square inch, respectively.

The welded tube took a large deflection before the temperature had reached 600° C and failed to sustain full load. A slight application of water increased the deflection and caused it to fall out of the machine. The two other columns were heated to 300 and 400° C, followed in each case by water application, their behavior being similar to that of the cast-iron columns, only the deflections were larger. On heating to 600° C and applying water the deflections became so large that the full load could not be carried, and some of the bolts were sheared off.

Of the tests on other building materials that with Portland-cement mortar was made on a column about 12 inches (30 cm) square and 10 feet (3 m) long. The proportions of the mixture was 1:5 Portland cement and coarse sand. The column was tested at the age of 6½ months under a working load of 95 pounds per square inch. It was heated for 1¾ hours until a temperature of 600° C was attained at the middle of the sides, when water was applied. No apparent injury resulted, and when cold it was loaded to failure at 920 pounds per square inch.

The brick column was about 12½ inches (32 cm) square and 6.6 feet long with brick laid in Portland-cement mortar and the outside covered with a layer of Roman-cement plaster about 0.60 inch (1.5 cm) thick. It was exposed to fire for one hour attaining a temperature of 600° C on the sides, when water was applied. No apparent damage other than slight cracking and flaking of the plaster resulted, the column being subsequently loaded to failure at load of 520 pounds per square inch.

(*b*) Second Series.—Objections raised against Bauschinger's tests, in particular that the loads applied to the cast iron-columns were too small and that the component

parts of the wrought-iron columns were bolted together at intervals too far apart to secure a rigid section led Bauschinger to conduct a second series of tests in 1886 on two cast and five wrought iron columns. The cast-iron columns were about 13 feet (4 m) long; one had outside diameter of 7 inches and average wall thickness of 1.1 inch, and the other had outside diameter of 6.1 inch and average wall thickness of 1 inch, the applied working loads being 8400 and 7100 pounds per square inch, respectively. The columns had fairly uniform wall thickness and were apparently of better quality than the cast-iron columns of the first series. They were tested in a manner similar to those in the first series, and although large deflections occurred when heated to red heat and suddenly cooled by water application, they sustained their full working load. No cracks due to water application developed.

The wrought-iron columns were about 19 feet long and consisted of two plate and channel box sections and three starred-angle sections. The former were built up of two 5.7-inch channels and two $\frac{5}{16}$ by 8 inch (20 cm) plates with rivets spaced 3 to 5 inches on centers, and were loaded to 6300 pounds per square inch. The starred-angle sections were of two sizes, two being built up of four $3\frac{1}{2}$ by $3\frac{1}{2}$ by $\frac{7}{16}$ inch angles spaced $2\frac{1}{4}$ inches back to back and riveted together rigidly at the ends and at two intermediate points about 6 feet apart. The angles of the third column of this type had $3\frac{1}{8}$ by $3\frac{1}{8}$ by $\frac{3}{8}$ inch (1 cm) angles spaced 1.4 inches back to back and were riveted at the ends and at five intermediate points about 3 feet apart. The working loads for the two types were 6300 and 5300 pounds per square inch, respectively.

All five columns were heated red hot, and water applied until they were cold. The two box columns carried their full load throughout the test, although large permanent deflections were produced. The angle columns deflected downward rapidly and were unable to carry their full load at 600° C. Water application increased the deflection and caused a further decrease in the sustained load. The rivets were distorted, but none were sheared off

2. TESTS BY MÖLLER AND LÜHMANN [4]

In 1887, M. Möller, a Government architect, and R Lühmann, an engineer, obtained a prize offered by the (German) Society for the Promotion of Industrial Progress for the best essay on the subject of the "Resistance of Iron Columns when Subjected to High Temperatures," submitting to that end a paper in which were described tests conducted by them at Hamburg, Germany.

From the remarks accompanying the announcement of the competition it is evident that the principal interest centered in the relative resistance to fire and water of unprotected cast and wrought iron columns, although the behavior of masonry piers at high temperatures was also considered of importance.

In planning their investigation, Bauschinger's work was carefully considered by Möller and Lühmann, who concluded that in both the first and second series of tests by the former, the wrought-iron columns were overloaded, resulting in relatively greater bending as the columns deflected under the unsymmetrical heating and giving greater extreme fiber stress than the columns could withstand. It was held that a comparison of cast and wrought iron columns should be made with specimens identical in length, lateral dimensions, and moment of inertia.

The specimens tested by Möller and Lühmann were 3.28 feet (1 m), 6.56 feet (2 m), or 13.12 feet (4 m), in length. The shortest length was chosen to give approximately the compressive strength of the material, the intermediate length was in about the

[4] Verhandlung des Vereines zu Beforderung des Gewerbfleisses, 66, p. 573 and 701; 1887.

same ratio to the lateral dimensions as of columns in actual use, and the longest was taken as representing unusually large values of that ratio.

The cast and wrought iron specimens, all approximately 9.8 square inches in cross-sectional area, comprised solid and hollow cylindrical forms, the former 3½ inches in diameter, the latter about 6 inches in outer diameter with wall thickness slightly less than ⅝ of an inch; a hollow, cast-iron, ornamental, fluted column of approximately the same diameter and area as the other hollow sections; and a riveted form of rectangular section (6 by 7 inches) built up of four angles united on two sides by plates and on the other sides by lattice bars, the component parts being rigidly bolted and riveted together. The cast iron columns had fairly uniform wall thickness.

As adding somewhat to the resistance to fire and water, one of the wrought-iron and two of the cast-iron columns were filled with cement, a 2-inch gas pipe being placed concentrically in one of the latter to maintain alignment of the pieces when the column broke. Quite full protection was given one each of the hollow cast and wrought iron specimens by a covering of 1:3 cement mortar about 2½ inches thick, interlaced with wire. Also, one of the riveted angle columns was protected by a circular wood mantle about 1¼ inches thick, covered with sheet iron.

There were further included oak and fir columns 6 inches square and columns of brick 9 inches square set in cement mortar.

Lengths other than 6.56 feet were used only in the unprotected cylindrical cast and wrought iron specimens. Report was made on about 40 tests, of which approximately two-thrids were fire and water tests, the rest being load tests at normal temperature.

The testing apparatus was very similar to that used by Bauschinger, the columns being loaded in a horizontal hydraulic press and heat applied by a coke fire in a U-shaped trough placed beneath the column. Temperatures of 330, 440, and 600° C were measured by metals and alloys placed in contact with the outside of the columns. Vertical deflections were measured at the mid-point of the length, using a lever with one of its ends resting on the column.

The loads were applied with the column axis uniformly 0.39 inch (1 cm) below that of the machine, the authors holding that central loading of columns does not occur in buildings except by chance, and that the above eccentricity is about what may be expected in practice.

In general the tests were conducted with the column ends unrestrained, the machine being fitted with spherical bearing blocks, but to determine the difference a few columns were loaded between fixed parallel bearing plates. Some of the columns were loaded at ordinary temperatures, and the safe load, as judged apparently by the limit of permissible deflection, and the maximum load were determined. The same quantities were determined for the high temperature condition, in general after 330 and 600° C had been attained on the upper and lower sides of the column, respectively. The load determinations at high temperature were generally made when the maximum deflection had been induced, which occurred at the time of water application.

On application of maximum load while hot, the short hollow columns (1 m long) deflected upward due to yielding of the metal at higher temperature on the lower side of the column. The solid columns of this length, in common with all of the longer columns, deflected downward due to the uneven heating, the deflections increasing on application of water and load. The following tabulation gives a summary of the results obtained with unprotected hollow and solid columns of cast iron and wrought iron:

Form and material	Effective length	Slenderness, ratio $\frac{l}{r}$	Fire and water test		Test at normal temperature	
			Maximum load	Deflection	Maximum load	Deflection
	Feet		Lbs./in.²	In.	Lbs./in.²	In.
Hollow cast iron	4.2	27	16 500	0.98	43 700	1.02
Hollow wrought iron	4.2	27	13 500	1.97	.....	.....
Hollow cast iron	7.5	47	23 000	1.77	37 200	.39
Hollow wrought iron	7.5	47	10 800	1.97	23 000	.78
Solid cast iron	4.2	57	9220	1.17	39 300	.33
Solid wrought iron	4.2	60	9100	.28	22 100	.63
Solid cast iron	7.5	101	6640	1.93	10 900	.98
Solid wrought iron	7.5	107	5000	1.97	13 500	.18
Hollow cast iron	14.0	80	2650	4.75	15 300	1.77
Solid cast iron	14.0	190	1890	6.80	.....	.....
Solid wrought iron	14.0	201	1620	4.56	.....	.....

The decrease in strength with length was marked, particularly for the columns loaded after fire exposure and water application. This was due to the larger lateral deflections caused by unsymmetrical heating, cooling, and loading induced in the longer columns, test conditions that can hardly be said to duplicate typical fire conditions even for unprotected exterior wall columns. In the shorter lengths, the cast iron sustained higher average unit loads than the wrought iron, the difference becoming smaller with increase of length or slenderness ratio. The load reported as safely carried after fire and water application varied from a little over one-half of the maximum load finally sustained to nearly the full value of the latter. Failure was due in all cases to bending produced mainly by uneven temperature distribution over the column section. The tests developed no cracking of the hot cast iron due to water application except in the case of the longest column, in which case the crack could have been incidental to failure as caused by the large deflection.

The columns tested between fixed parallel plates withstood higher loads than those tested with spherical end blocks, although the tests were too few in number to afford definite comparison.

The filled columns sustained slightly higher loads under the same test conditions than the corresponding unfilled columns. The metal of the columns protected by mortar was only moderately warm after the column had been in the fire about 1½ hours, and that of the column protected by wood was 120° C after the covering had been removed subsequent to a 63-minute fire exposure. The strength of the brick columns was reduced about 50 per cent after 40 to 45 minutes' fire exposure. The oak column failed due to reduction of area and strength of the wood after 8 minutes under load of 1240 pounds per square inch, as computed on the original area, and the fir column after 18 minutes in the fire under load of 1110 pounds per square inch.

3. HAMBURG TESTS[6]

Two series of fire tests on loaded building columns were undertaken at Hamburg, Germany, in the period 1893 to 1895, by representatives appointed by the Hamburg

[6] Vergleichende Versuche uber die Feuersicherheit von Speicher-stutzen. Commissions--bericht Hamburg, 1895.
Vergleichende Ubersicht uber die Feuersicherheit gusseiserner Speicher-stutzen. Hamburg, 1897.
Zeitschrift d. Verein Deutch Ing., **40**, p. 159; **41**, p. 1007; **42**, p. 183.
Stahl und Eisen, **18**, p. 696.

senate from the building commission, the fire department, the warehouse association, and the insurance interests.

(*a*) First Series.—In the first series tests were made on wrought-iron columns built up of four angles united by latticed bars and plates to form rectangular sections. A few columns were tested unprotected, two were filled with cinder concrete, and nine were protected by coverings from ¾ to 2 inches in thickness, consisting of concrete (Monier) blocks, gypsum and magnesite blocks or boards, pressed cork, and asbestos cement. Some of the coverings were protected by a sheet-metal mantle. In the series were also included three wooden columns 11.8 inch square, one of which was covered by sheet metal.

The question of the relative fire resistance of the several column materials was still of interest, and, although fire-protective coverings had come to be employed, their use had not gained such headway as in America, where hollow tile, at that date not manufactured in Germany, was the almost universal covering material. The importance of simulating in the tests actual loading and fire conditions was recognized.

The columns were loaded by a hydraulic ram in upright position within a framework in which were inserted two platforms 11½ feet apart, to which the test column was attached. Heat was applied over a length of about 4 feet in the middle portion of the column by a gas-fired oven built in halves which swung together about the column. Temperatures were measured by Seger cones, fusible alloys, and a thermoelectric pyrometer, and lateral deflections with rods passed through holes in the furnace walls. Working load was applied, centrally or eccentrically, and heating continued until the column was unable to sustain it, when water was applied over the heated length. The furnace temperature rise varied considerably between different tests and was at times quite rapid, 1200° C being sometimes attained in 2 hours and 1400° C in 4 hours.

The unprotected wrought-iron columns of the first series failed after 17 to 59 minutes' fire exposure, depending on the rate of temperature rise, the metal temperature at failure being given at about 600° C, and the load sustained 14 200 pounds per square inch (1000 kg per sq. cm). The filled columns proved to be only slightly more resistive than the protected and unfilled columns. The protected wrought-iron columns failed after 1¾ to 4 hours, with maximum furnace temperature of 1200 to 1300° C, the load applied being apparently the same as for the unprotected columns. The temperatures inside of the column were not known, except that they were over 412° C.

The application of water resulted in injury to the concrete coverings, and distruction of the other coverings, except where covered with sheet metal.

The wooden columns failed after a furnace exposure averaging a little over 1 hour for the three tests, the reduction in cross section from combustion of the wood being about 40 per cent.

(*b*) Second Series.—Tests were made on about 24 columns of cast iron of 10.8 inch outside diameter, 22 of which had wall thicknesses of 1.18 inch, and two 0.47 inch, the columns being cast in vertical position. Of the total number 17 were protected by coverings 1½ to 2 inches (4 to 5 cm) thick consisting of the materials used in the first series, to which were added coverings of tufa stone and asbestos-

kieselguhr. A number of the coverings were encased in sheet metal. The effect of an air space between the covering and column and of free convection within the column were studied, as, also, the difference between permanent as distinguished from removable coverings, some members of the commission holding that coverings should be removable to permit examination of the column from time to time on account of danger from rust.

The method of testing was substantially the same as in the first series. The unprotected cast-iron columns failed in from 35 to 59 minutes, depending on the intensity of the fire, under load of 7100 pounds per square inch (500 kg per sq cm), and temperature of the column reported as 800° C. Increasing the column load to 10 700 pounds per square inch (750 kg per sq cm) caused failure to occur with column temperature of 700° C. The thin-walled column gave a little lower resistance than those of greater thickness. The application of water to unprotected columns that had failed in the fire test and suffered large deformation generally caused cracking.

The protected columns failed after 3 to 5½ hours, with maximum furnace temperatures from 1200 to 1500° C, except the one covered with 2-inch thickness of asbestos-kielselguhr, which did not fail after a fire exposure of 7 hours. The air space between the column and the covering increased the fire resistance a little and the provision for free convection within the column delayed failure by nearly one hour. The action of water on the coverings was the same as in the first series.

(*c*) General Results.—The Hamburg tests were the first to determine the strength of wrought iron and cast iron, applied as columns, under central load and symmetrical heating. The column temperatures reported, regarded as those of the metal at failure, were without doubt higher than the actual temperatures, due to the methods of measurement employed. As column tests they are open to objection in that only about one-third of the column length was heated. The tests on protected columns while giving little information on the effectiveness of covering materials now in use, proved conclusively the great gain in fire resistance attainable by protecting the metal with materials of low heat conductivity.

With regard to water application on cast-iron columns at high temperature, the results are not conclusive, since in almost all tests the columns had sustained large deformations by failure in the fire test before water was applied.

4. FIRE TEST OF COLUMN AT VIENNA [6]

In 1893 the Building Department of Vienna conduced a fire and water test on a single wrought-iron column 11½ feet long, built up of two 5½-inch channels connected by lattice bars. The column was protected by brickwork 5½ inches thick, laid in fire clay mortar, the interior of the column being unfilled.

The furnace was of brick 8 by 12 feet, and with height equal to the column length. Wood piled 3 to 4 feet high was used for fuel, and the temperatures of furnace and column were indicated by alloys. Load was applied to the column section, using a long lever. After a fire test of 2½ hours duration, wherein furnace temperatures evidently much in excess of 400° C were attained, a hose stream was applied to one side of the column.

The only apparent damage caused by the combined test was considerable spalling of the bricks at the corners, cracking of bricks in the upper half of the covering, and washing away of some of the mortar. The maximum temperature indicated inside of the column was 65° C.

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[6] Engineering News, **32**, p. 184; Sept. 6, 1894. Translation of article in Zeitschrift des Oestereicheschen Ingen. u. Arch. Verein.

5. NEW YORK TESTS [7]

Fire tests of unprotected columns, two of structural steel and two of cast iron, and a fire and water test of one unprotected cast-iron column, were made in 1896 under the direction of a committee appointed by the Tariff Association of New York, the Architectural League of New York, and the American Society of Mechanical Engineers.

The steel columns were 14 feet long, one being a box section built up of two 10-inch channels and two 12 by ¼ inch plates, and the other a Z-bar section consisting of four 4 by $\frac{5}{16}$ inch Z-bars riveted to a plate 6½ inches wide, other details of design being in accord with accepted standards of practice. The cast-iron columns were 13 feet long, of 8-inch outside diameter and 1-inch wall thickness, and were cast in horizontal position with dry-sand core, the ends being flanged and end bearings machined.

The columns were tested in vertical position within a furnace chamber about 12 feet square and 14 feet high, the fuel used being producer gas admitted by burners extending through the floor of the furnace. Air was supplied through openings in the floor near the gas inlets. Arrangements were made for intensifying the fire when necessary by injecting a naphtha spray into the gas main supplying the burners. Load was applied to the column within a restraining frame of structural steel by a hydraulic ram placed beneath the column, filler blocks of cast iron transmitting the load to the column.

In the fire tests it was intended to subject the columns to working loads, and for one test each of steel and cast-iron columns to use a slow and a rapid furnace temperature rise. Furnace temperatures were measured with an Uhling-Steinbart transpiration pyrometer. Temperatures on the metal of the columns were not measured in any of the tests.

In the test of the plate and channel column, using a slow temperature rise, a furnace temperature of about 650° C was attained in one hour, which temperature maintained for about 20 minutes caused the steel to show red color and fail under a load of 6400 pounds per square inch, trouble with the loading equipment preventing the application of the full working load.

The Z bar column, tested under full working load of 12 000 pounds per square inch and rapid furnace temperature rise, failed 25 minutes after the beginning of the test. A maximum furnace temperature of about 750° C was indicated at 14 minutes, which fell off to 600° C at failure.

The cast-iron columns were tested under working load of 7700 pounds per square inch. In the test with slow temperature rise the column began to show color after 65 minutes' fire exposure, the furnace temperature being near 600° C. After another 18 minutes, during which the furnace temperature averaged 640° C, the gas was shut off and the furnace door opened for 9 minutes, which disclosed the test column to be decidedly red and bent. The door was closed and the test continued at lower furnace temperature for 28 minutes, the column sustaining its load, although greatly bent. No cracks developed. The permanent lateral deflection was 3½ inches.

The cast-iron column tested with rapid temperature rise had deflected a visible amount at 35 minutes and began to show color at 39 minutes, the furnace temperature at this time being 730° C. It failed at 43 minutes with complete fracture across the section near the middle.

In the fire and water test, water was applied at four successive times following fire exposures of increasing intensity. Before the third application the furnace temperature was 580° C, and the column showed color. Immediately before the last water

[7] Trans. Am. Soc. of Mech. Eng., 18, p. 24, 1896–1897.

application a furnace temperature of 700° C was indicated, and the column was red and bent. No cracks developed. The permanent lateral deflection was 3¼ inches.

The tests made apparent that unprotected metal columns will fail in fires of moderate intensity after a comparatively short exposure, the steel columns being less resistive than those of cast iron under the respective unit loads for which the two types are generally designed. Injury to cast-iron columns due to water application while hot is shown to be improbable with columns of the given proportions. The tests give little information on the temperature in the metal at failure or on definite time resistance under comparable fire conditions.

6. WAITE'S TESTS

In 1903 the Guy B. Waite Co., of New York, conducted some fire and water tests on floors and partitions, under the supervision of the Manhattan Bureau of Buildings, constructing to that end a reinforced-concrete test house, in which were placed, for the purpose of studying incidentally the behavior of column coverings, a reinforced-concrete column made with stone aggregate, proportion of mixture, 1 : 2 : 4, and two cast-iron columns, one of which was covered with 1½ inches of poured cinder concrete of proportion 1 : 5, Portland cement, and hard-coal cinders, the other with a gypsum and cinder composition.

The reinforced-concrete column was considerably pitted by water from a 1⅛-inch nozzle at 60 pounds' pressure after it had been subjected for 4 hours to a test fire with average temperature of about 930° C. The cinder-concrete covering was in excellent condition after it had gone through the same test, and, in addition, three tests of 1-hour duration with water application at the end of each. The gypsum covering appeared in good condition at the end of a single fire test, but on water application it was partially washed away.

7. TESTS BY McFARLAND AND JOHNSON[8]

Tests to determine the effect of fire and water on the strength of reinforced-concrete columns were made in 1906 by H. B. McFarland and E. V. Johnson at the Chicago laboratory of the National Fireproofing Co.

The columns, three in number, about 10½ inches square and 12 feet long, were made of 1 : 2 : 4 limestone concrete and reinforced with ⅜-inch rods placed near the corners. Two columns were tested in compression at normal temperature at age of 2 months and 3½ months, respectively, and sections 5 to 6 feet long cut from them outside of the region of failure for use in the fire test. One of these specimens was covered with 3-inch solid porous clay tile and the other was tested unprotected, and after the fire test subjected to water application. The third column was cut in two, one part for use in the fire test and the other for a comparable compression test at normal temperature. The age of all columns was 23 months at the time of the fire test.

The three specimens were placed on end in a wood-fired furnace and subjected under no load to a 3-hour fire test. Furnace temperatures, as indicated by a Bristol thermoelectric pyrometer, ranged from 800 to 1000° C for the greater portion of the period. On the day following the fire test they were tested in compression.

The section protected by clay tile was little affected by the test and developed compressive strength of 3127 pounds per square inch. An 18-inch long specimen cut from the same column but not subjected to fire test developed 3558 pounds per square inch. The specimen to which water was applied after the fire test failed in the compression test at 674 pounds per square inch. The column from which it was cut had

[8] Engineering News, 56, p. 316; Sept. 20, 1906.

developed 2116 pounds per square inch at age of 3½ months, the fire and water treatment having apparently caused a decided decrease in strength. A similar effect was indicated in the case of the third specimen, its strength being 711 pounds per square inch, against 2565 pounds per square inch for the section of the same column that was not exposed to fire.

The large reduction in strength sustained by the unprotected columns can be ascribed, in part, to their small size, larger columns being subject to smaller percentage reduction in strength due to surface damage from fire exposure.

WASHINGTON, June 4, 1920.

APPENDIX F

CENTIGRADE AND FAHRENHEIT CONVERSION TABLE

C.°	0	10	20	30	40	50	60	70	80	90
	F.°	F.°	F.°	F.°	F.°	F.°	F.°	F.°	F.°	F.°
0	32	50	68	86	104	122	140	158	176	194
100	212	230	248	266	284	302	320	338	356	374
200	392	410	428	446	464	482	500	518	536	554
300	572	590	608	626	644	662	680	698	716	734
400	752	770	788	806	824	842	860	878	896	914
500	932	950	968	986	1004	1022	1040	1058	1076	1094
600	1112	1130	1148	1166	1184	1202	1220	1238	1256	1274
700	1292	1310	1328	1346	1364	1382	1400	1418	1436	1454
800	1472	1490	1508	1526	1544	1562	1580	1598	1616	1634
900	1652	1670	1688	1706	1724	1742	1760	1778	1796	1814
1000	1832	1850	1868	1886	1904	1922	1940	1958	1976	1994
1100	2012	2030	2048	2066	2084	2102	2120	2138	2156	2174
1200	2192	2210	2228	2246	2264	2282	2300	2318	2336	2354
1300	2372	2390	2408	2426	2444	2462	2480	2498	2516	2534
1400	2552	2570	2588	2606	2624	2642	2660	2678	2696	2714
1500	2732	2750	2768	2786	2804	2822	2840	2858	2876	2894
1600	2912	2930	2948	2966	2984	3002	3020	3038	3056	3074
1700	3092	3110	3128	3146	3164	3182	3200	3218	3236	3254
1800	3272	3290	3308	3326	3344	3362	3380	3398	3416	3434
1900	3452	3470	3488	3506	3524	3542	3560	3578	3596	3614
2000	3632	3650	3668	3686	3704	3722	3740	3758	3776	3794

C.°	F.°
1	1.8
2	3.6
3	5.4
4	7.2
5	9.0
6	10.8
7	12.6
8	14.4
9	16.2
10	18.0

F.°	C.°
1	.56
2	1.11
3	1.67
4	2.22
5	2.78
6	3.33
7	3.89
8	4.44
9	5.00
10	5.56
11	6.11
12	6.67
13	7.22
14	7.78
15	8.33
16	8.89
17	9.44
18	10.00

Examples:

515° C.= 950° F.+9° F.= 959° F.
2039° F.=1110° C.+5° C.=1115° C.

DEPARTMENT OF COMMERCE

TECHNOLOGIC PAPERS

OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 185

EXPERIMENTS ON COPPER CRUSHER CYLINDERS

BY

ALEXANDER I. KRYNITSKY, Associate Physicist

Bureau of Standards

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EXPERIMENTS ON COPPER CRUSHER CYLINDERS

By Alexander I. Krynitsky

ABSTRACT

The experiments consisted of compression tests on a standard Riehlé testing machine. The results of these compressions carried out on annealed copper cylinders, one set having a mean length of 0.4000 inch and the others 0.5000 inch, with a mean diameter of 0.2260 inch in the former and 0.2500 and 0.2250 inch in the latter ones, suggest that the length of the cylinders decreased considerably under repeated application of the same and, within certain limits, of smaller loads.

In the case where the load is applied in such a manner as to produce the maximum stress for only an instant, the duration of an application of load has but little effect on the decrease in length of the cylinders, but the application of the same load by holding the beam balanced through an additional application of this load causes about twice the decrease in length as compared with that obtained by a single application of load. When two successive loads of considerable amount differing by increments of about 2000 pounds per square inch are applied, the second being greater than the first, the change in length due to the last load is greater than that obtained where the pressure is applied on previously uncompressed cylinders, and this difference decreases as the difference between the two loads successively applied increases, so that when the difference between the loads reaches a certain value the change of length due to the last load applied is practically independent of the previous load.

It appears probable that aging at temperatures within the range 0–100° C softens the compressed copper somewhat.

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I. INTRODUCTION

1. TWO METHODS FOR MEASUREMENT OF PRESSURE OF GASES IN TESTING OF POWDER AND AMMUNITION

Among many methods and apparatus [1] proposed for the measurement of pressure developed by the products of decomposition of powder in the testing of powder and ammunition there are two methods which up to the present time are commonly employed in practice; that is (1) the so-called dynamic method and (2) the method of measuring pressure by means of metallic cylinders, usually copper or lead cylinders, known as crusher cylinders.

The dynamic method consists of determining the velocities of the gun in recoil or of the shot at different points of the bore. The difference of the velocities divided by the corresponding differences of the time equal the acceleration. The pressure required to produce the observed acceleration in a body whose mass is that of the gun or of the projectile is obtained by multiplying the acceleration by the mass.

The crusher-gage method was proposed in 1868 by Noble, and up to the present time is regarded as the most convenient known method for obtaining an idea of the relative values of pressures in guns.

There is a metal cylinder set on an anvil. On firing, the pressure of the gases shortens the cylinder longitudinally. The amount of this compression, with a previously prepared table, so-called tarage table, serves to determine the maximum pressure of the products of decomposition. For the testing of small arms a special barrel is used. For example, in Figs. 1 and 2, a pressure-gage outfit

[1] It may be mentioned here that two types of registering apparatus proposed very recently appear very promising, and they possibly will find common use in the near future. One of these was proposed by Prof. A. G. Webster, of Clark University, and the other is designed to coordinate time and pressure within a large cannon, upon which Dr. H. L. Curtis and associates, of the electrical division of the Bureau of Standards, are working.

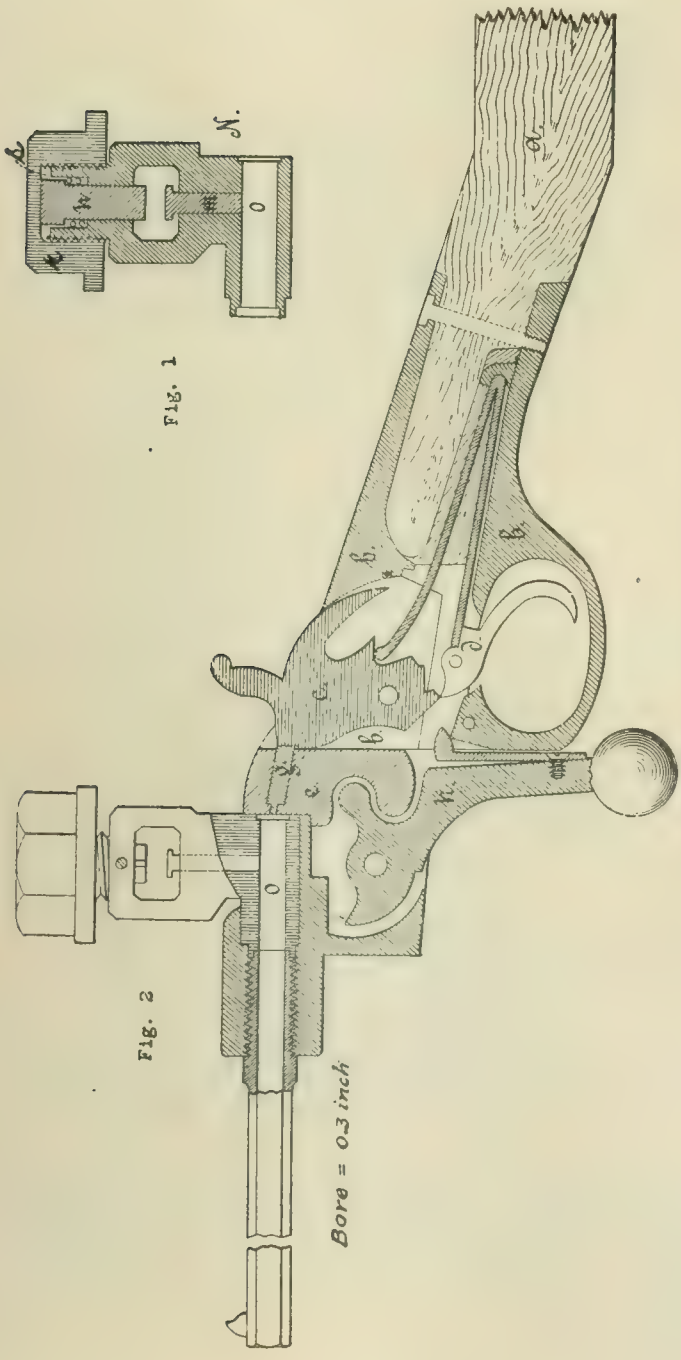

FIG. 1.—*Cross section of the pressure-gage attachment of Fig. 2*
FIG. 2.—*A pressure-gage outfit for Russian 0.3-inch revolver*

for Russian 0.3-inch revolver is shown: O = a special barrel; h = an unmovable piston fixed rigidly in its position by means of nut r and spring s; m = a movable piston. Between pistons h and m crusher cylinders are placed. By turning the nut r the crusher cylinder may be tightened between these two pistons.

2. COPPER CRUSHER CYLINDERS

In the present paper only copper crusher cylinders will be considered. Dimensions of copper cylinders vary in different countries and for pressure testing of different grades of ammunition and powder.

For example, some dimensions of copper crushers are as follows:

Dimensions	A	B	C	D
	mm	mm	Inch	Inch
Diameter	3	8	0.2260	0.2520
Height	4.9	13	.4000	.5000

The copper must be exceptionally pure, and the whole process of manufacture of copper crushers should be standardized with extreme care.

3. PREPARING DATA FOR TARAGE TABLE

In preparing these data the copper cylinders are slowly compressed by applying the loads and measuring the amount of compression at each load. The load for each compression remains active for about 15 seconds. Ch. B. Wheeler,[2] from his experiments, found it might safely be assumed that at the end of 15 seconds the copper is practically in equilibrium with its load and that any further application of load produces no effect.

Influence of the friction of the testing machine is of great importance in the construction of tarage tables. As early as 1891 this was demonstrated by Vieille, who showed that neglect of this factor was responsible for the erroneous French crusher tables.

This point, states F. W. Jones,[3] must be insisted upon because experiments in some quarters assume friction is of no account, a conclusion to which they are driven because no provision has been made in the design of their testing machine for applying the load with friction acting alternately back and forth.

[2] Report of Experiments with an 8-inch B. L. Rifle Mounted on a Free Recoil Carriage, by Lieut. Ch. B. Wheeler, Ordnance Construction, Notes 69–85, pp. 46–56; Jan., 1896.

[3] A Standard Pressure Table for Copper Crushers, by F. W. Jones, Arms and Explosives, June 1, 1918.

4. COMPARISON OF PRESSURES INDICATED BY DYNAMIC AND CRUSHER-GAGE METHODS

The dynamic method shows a pressure which is less than the true pressure expected in the gun, as it takes no account of the pressure required to overcome the friction of the projectile in the bore and to rotate the projectile.

Comparing now the dynamic method and the crusher-gage method, we find that the latter gives pressures which are less than those shown by the dynamic method.

According to B. W. Dunn,[4] under ordinary service conditions (since the time required by powder pressure to reach a maximum is less than 0.003 second or about 0.001 second) the probable errors with the crusher-gage method will be greater than 5000 pounds per square inch.

The quicker the powder the greater is the excess of actual over indicated pressures and, for this reason, the error of the crusher gage is greater for small arms than for cannon.

5. NECESSITY OF PRELIMINARY PRECOMPRESSION

Though the crusher-gage method is very old, there remains up to the present time a problem of great importance—the necessity of precompression. It has been stated by many experimenters that copper cylinders should be precompressed at a certain pressure before using for test. Capt. Blunt,[5] F. W. Jones,[6] Marshall,[7] and others state that copper should be compressed beforehand with a static pressure only slightly less than that which it is to undergo in the actual test.

Col. W. H. Tschappat[8] states that when the pressure in the gun is high and quickly developed, it is at first very much greater than the resistance offered by the copper cylinder. The excess of pressure over resistance in this case accelerates the mass of the piston and develops considerable velocity in it. The energy of the piston due to this velocity is finally absorbed by further compression of the copper; but in the early stages of compresssion the mass of the piston prevents as great a compression as would be

[4] Report of Development of a Photoretardograph and Its Application to the Dynamic Measurement of Resistance to Compression Offered by the Copper Cylinders Used in Crusher Gages, by Lieut. B. W. Dunn, Ordnance Construction Notes, pp. 69–85; Jan. 13, 1896.

[5] Report of Capt. Blunt, Report of Chief of Ordnance, pp. 929–943; 1892.

[6] F. W. Jones, Lectures to Young Gunmakers, Arms and Explosives, p. 58, April, 1901, and p. 139, March, 1912.

[7] Marshall, Explosives, p. 363.

[8] Lieut. Col. W. H. Tschappat, Ordnance and Gunnery, pp. 101–103; 1917.

obtained if the pressure acted directly upon the copper cylinder instead of through the piston.

It is therefore desirable to reduce the energy of the piston as much as possible, and this may be done in two ways: By reducing its weight and by limiting its travel and accompanying velocity. As far back as 1883 Sarrau and Vieille [9] [10] showed that deformation of copper cylinders depends upon the weight (mass) of the piston through which the pressure is communicated. Usually the piston is made as light as possible.

To limit the travel all copper cylinders are precompressed by a load, generally about 3000 pounds per square inch less than that expected in the gun. According to many statements, initially compressed cylinders always record more uniform results than the uncompressed cylinders do.

Many investigators are of the opinion that the influence of many factors which have an important bearing on the results of pressure testing are minimized by the use of initially compressed cylinders, and they believe also that it is to their advantage to check each cylinder by an initial compression. On the other hand, there is rather good evidence that very uniform results may be obtained with uncompressed cylinders when they are made extremely uniform.

Thus, as was said above, the necessity of precompression up to the present time is rather an open question. In order to obtain some more data for the solution of this problem, experiments were started by the Bureau of Standards in connection with the standardization of the crusher-gage method for pressure testing of small arms, ammunition, and powder.[11] So far, the experiments have been confined to static tests; these are described in this paper. The experiments were carried out in 1919; they consisted of a series of compressions of the cylinders on a standard 10 000-pound Riehlé Brothers testing machine, and some microscopic examinations of their longitudinal and cross sections. The results are given in the tables and diagrams, so that few explanatory notes are required.

[9] Sarrau et Vieille, Etude sur l'Emploi des Manomètres pour la Mesure des Pressions Développées par le Substances Explosives.

[10] The Interior Ballistic, by Col. Pashkevitch, translated from Russian by Lieut. Tasker H. Bliss.

[11] The standardization of the crusher-gage method was started by Dr. P. D. Merica under the supervision of Dr. G. K. Burgess.

II. COMPRESSION TESTS

1. METHODS OF COMPRESSION

The rate of compression was always about 0.07 inch per minute. Duration of load after balance was obtained in most cases was about 2½ seconds, during which time the beam was not falling down.

Two methods of applying the load were used:

(1) There was single application of load. As soon as balance was obtained, the load was removed; or in some cases the load was removed after a certain period—5 or 25 seconds.

(2) After the balance was obtained, which was always followed by drop of the beam about 3 seconds after original balance, the balance of beam was maintained by reapplying the same load.

2. MEASUREMENT

The measurements were then made with a 1/10 000-inch micrometer. Measurements of length were taken with the cylinders placed in the micrometer centrally, each length being measured several times and only the minimum length recorded. Diameters were measured several times and the average recorded. After compression the cylinders have a barrel shape and in some cases two diameters of each cylinder were measured; that is, diameter of base; also diameter of cross section midway between the bases.

The diameter of the cross section midway between the bases is called "maximum" in the tables, and, as was stated, it is an average value of several measurements. The diameter of the base is called "minimum" and is also an average of several measurements. The minimum diameter, because of the difficulty of obtaining the exact measurement, is only an approximate value.

3. PLAN OF DESIGNATION

There are two lots of copper crusher cylinders both annealed at 1200° F at the same plant, but at different times. These two lots are here called Series No. 1 and Series No. 2.

There are also two other sets annealed at another factory at 1000° F and 1200° F, respectively, and called Series No. 3. The dimensions are given in inches, time in seconds, weight in pounds. Certain figures omitted were disregarded in drawing up the average as doubtful for various reasons. The following symbols will be used:

L = initial length of cylinders.

D = initial diameter of cylinders.

A = initial cross-sectional area.

L_1 = length of cylinders after the first compression.

L_2 = length of cylinders after the second compression, and so on.

D_1 = diameter of cylinders after the first compression.

D_2 = diameter of cylinders after the second compression, and so on.

$L - L_1$ = change in length of cylinders after first compression.

$L - L_2$ = change in length of cylinders after the second compression, and so on.

4. RESULTS OF COMPRESSION TESTS

(a) EXPERIMENTS ON REPEATED APPLICATION OF SAME AND OF SMALLER LOADS

TABLE 1.—Experiments on Repeated Application of Same and of Smaller Loads

L_1 = length of cylinders after applying pressure of 30 000 lbs./in.² for about 30 seconds.
L_2 = length of cylinders after applying pressure of 40 000 lbs./in.² for about 30 seconds.
L_3 = length of cylinders after applying pressure of (Note a) and (Note b) lbs./in.² for about 30 seconds.
L_4 = length of cylinders after applying pressure of 38 000 lbs./in.² for about 2½ seconds.
L_5 = length of cylinders after applying pressure of 36 000 lbs./in.² for about 2½ seconds.
L_6 = length of cylinders after applying pressure of 34 000 lbs./in.² for about 2½ seconds.
L_7 = length of cylinders after applying pressure of 30 000 lbs./in.² for about 2½ seconds.
L_8 = length of cylinders after applying pressure of 25 000 lbs./in.² for about 2½ seconds.

No.	L	D	A	Load equivalent to 30 000 lbs./in.²	$L-L_1$	D_1 maximum	Load equivalent to 40 000 lbs./in.²	$L-L_2$	D_2 maximum	Load equivalent to—(See [a] or [b])	$L-L_3$	Load equivalent to 38 000 lbs./in.²
	Inch	Inch	Inch²	Lbs.	Inch	Inch	Lbs.	Inch	Inch	Lbs.	Inch	Lbs.
1	0.5004	0.2514	0.0495	1485	0.0494	0.2672	1980	0.0814	0.2780	[a] 1980	.....	1881
2	.5009	.2516	.0497	1491	.0490	.2675	1988	.0786	.2778	[a] 1988	.....	1888
3	.5003	.2516	.0497	1491	.0484	.2675	1988	.0786	.2768	[a] 1988	.....	1888
4	.5006	.2517	.0497	1491	.0485	.2678	1988	.0802	.2781	[a] 1988	.....	1888
5	.5004	.2514	.0495	1485	.0487	.2669	1980	.0786	.2772	[a] 1980	.....	1881
6	.4996	.2516	.0497	1491	.0482	.2673	1988	.0799	.2784	[b] 1888	0.0816	1888
7	.5009	.2517	.0497	1491	.0482	.2677	1988	.0799	.2780	[b] 1888	.0810	1888
8	.5010	.2517	.0497	1491	.0501	.2673	1988	.0794	.2777	[b] 1888	.0823	1888
9	.5013	.2518	.0497	1491	.0480	.2672	1988	.0786	.2774	[b] 1888	.0808	1888
10	.5003	.2517	.0497	1491	.0492	.2673	1988	.0798	.2777	[b] 1888	.0820	1888
Average	.....	.....	.....	.....	.0488	.....	.....	.0795	.....	.....	.0815	.....

No.	$L-L_4$	D_4 maximum	D_4 minimum	Load equivalent to 36 000 lbs./in.²	$L-L_5$	Load equivalent to 34 000 lbs./in.²	$L-L_6$	Load equivalent to 30 000 lbs./in.²	D_7 maximum	D_7 minimum	$L-L_7$	Load equivalent to 25 000 lbs./in.²	$L-L_8$
	Inch	Inch	Inch	Lbs.	Inch	Lbs.	Inch	Lbs.	Inch	Inch	Inch	Lbs.	Inch
1	.....	0.2790	0.2669	1782	.....	1683	.....	1485	0.2795	0.2679	.....	1237	.....
2	.....	.2790	.2669	1789	.....	1690	.....	1491	.2795	.2675	.....	1243	.....
3	.....	.2780	.2660	1789	.....	1690	.....	1491	.2785	.2675	.....	1243	.....
4	.....	.2800	.2645	1789	.....	1690	.....	1491	.2805	.2675	.....	1243	.....
5	.....	.2790	.2660	1782	.....	1683	.....	1485	.2790	.2675	.....	1237	.....
6	0.0837	.2790	.2650	1789	0.0842	1690	0.0861	1491	.2796	.2675	0.0867	1243	0.0867
7	.0842	.2795	.2650	1789	.0851	1690	.0861	1491	.2788	.2675	.0865	1243	.0872
8	.0843	.2790	.2669	1789	.0850	1690	.0859	1491	.2798	.2675	.0863	1243	.0864
9	.0835	.2780	.2650	1789	.0862	1690	.0877	1491	.2800	.2675	.0880	1243	.0880
10	.0835	.2785	.2660	1789	.0848	1690	.0862	1491	.2795	.2675	.0866	1243	.0868
Av.	.0838	.....	.....	.....	.0851	.....	.0865	.....	.....	.....	.0868	.....	.0870

[a] Load in pounds equivalent to 40 000 lbs./in.² [b] Load in pounds equivalent to 38 000 lbs./in.²

NOTES ON TABLE 1.—Experiments were made upon copper crusher cylinders of Series I. The first and second compressions were made January 31, 1919. The third compression was made February 13, 1919. All three compressions were made under condition (2), stated above, keeping beam balanced for about 30 seconds after balance was obtained. The rest of the compressions

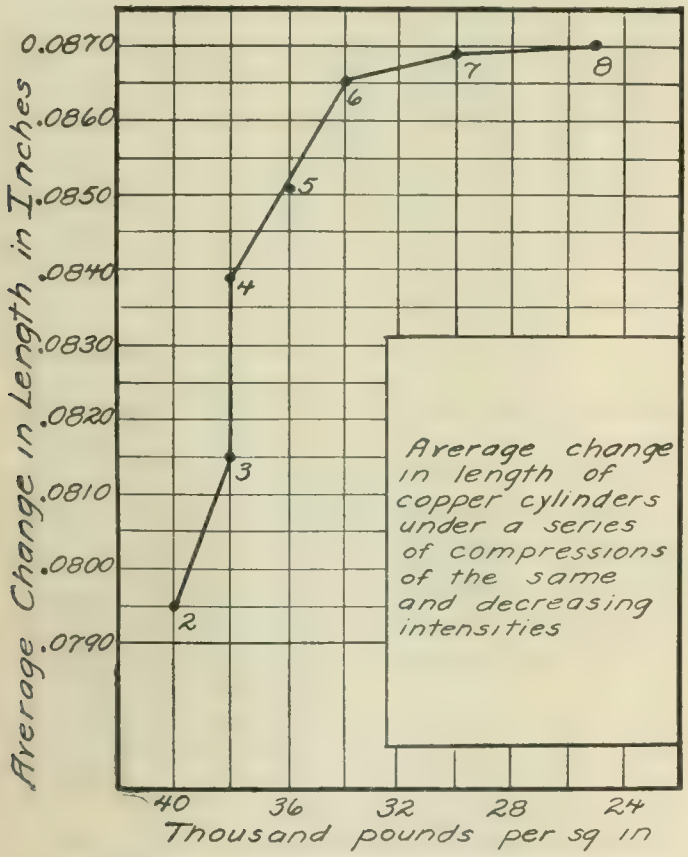

FIG. 3.—*Change of length of copper cylinders upon compression*

(compressions 4–8) were made May 17–19, 1919, under condition (1), with single application of load for about 2½ seconds.

As is shown in Fig. 3, the length of the copper cylinders decreases considerably under repeated application of the same and even smaller loads; this is in spite of the fact that the first three compressions were made keeping the beam in a balanced position for about 30 seconds.

TABLE 2.—Experiments on Repeated Application of Decreasing Loads

L_1 = length of cylinders after applying pressure of 30 000 lbs./in.² for about 30 seconds.
L_2 = length of cylinders after applying pressure of 28 000 lbs./in.² for about 2½ seconds.
L_3 = length of cylinders after applying pressure of 25 000 lbs./in.² for about 2½ seconds.

No.	L	D	A	Load equivalent to 30 000 lbs./in.²	$L-L_1$	D_1 maximum	Load equivalent to 28 000 lbs./in.²	$L-L_2$	Load equivalent to 25 000 lbs./in.²	L_3	$L-L_3$
	Inch	Inch	Inch²	Pounds	Inch	Inch	Pounds	Inch	Pounds	Inch	Inch
1	0.5010	0.2517	0.0497	1491	0.0486	0.2674	1391	0.0491	1243	0.4515	0.0495
2	.5001	.2517	.0497	1491	.0496	.2675	1391	.0505	1243	.4496	.0505
3	.4999	.2515	.0496	1488	.0482	.2671	1388	.0494	1240	.4502	.0497
4	.5010	.2517	.0497	1491	.0475	.2668	1391	.0496	1243	.4500	.0510
5	.5006	.2515	.0496	1488	.0481	.2670	1388	.0507	1240	.4499	.0507
6	.5002	.2515	.0496	1488	.0482	.2670	1388	.0505	1240	.4487	.0515
7	.4998	.2518	.0497	1491	.0468	.2663	1391	.0483	1243	.4505	.0493
8	.5000	.2516	.0497	1491	.0468	.2668	1391	.0484	1243	.4505	.0495
9	.5005	.2515	.0496	1488	.0467	.2677	1388	.0484	1240	.4503	.0502
10	.5003	.2514	.0495	1485	.0482	.2665	1386	.0501	1237	.4493	.0510
Average					.0478			.0495			.0503

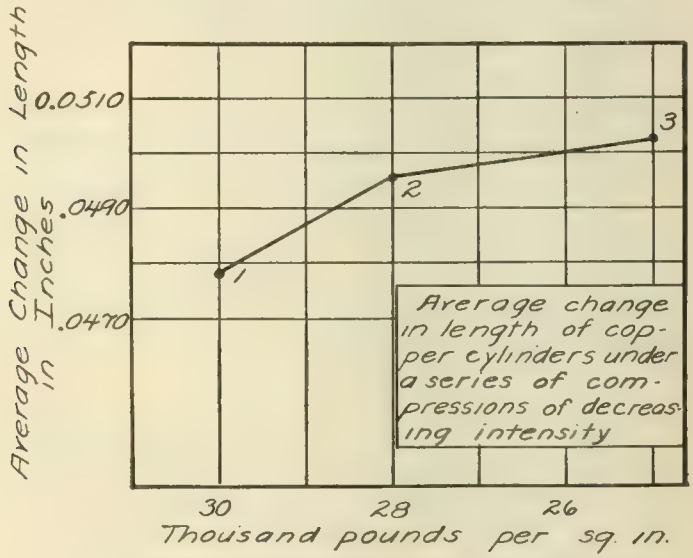

FIG. 4.—*Change of length of copper cylinders upon compression*

NOTES ON TABLE 2.—Experiments were made with copper cylinders of same stock as those of Table No. 1. The first compression was made February 1, 1919, with load of 30 000 pounds per square inch applied under condition (2), keeping the beam balanced for about 30 seconds.

The second and third compressions were made May 20, 1919, with single application of load 28 000 and 25 000 pounds per square inch for about 2½ seconds under condition (1). Fig. 4 indicates that repeated application of decreasing loads gradually decreases length of cylinders (within certain limits).

TABLE 3.—Experiments on Repeated Application of Same Load

No.	L	D	A	Load equivalent to 40 000 lbs./in.²	D_1 maximum	D_1 minimum	$L-L_1$	$L-L_2$	$L-L_3$
	Inch	Inch	Inch $_2$	Pounds	Inch	Inch	Inch	Inch	Inch
1	0.4010	0.2265	0.04029	1612	0.2500	0.2350	0.0618	.....	.....
2	.4008	.2260	.04012	1604	.2485	.2350	.0586	0.0616	0.0638
3	.4005	.2256	.03997	1599	.2490	.2350	.0608	.0645	.0658
4	.4004	.2261	.04015	1606	.2468	.2350	.0547	.....	.....
5	.4005	.2262	.04018	1607	.2483	.2350	.0579	.0614	.0640
6	.4002	.2267	.04036	1614	.2498	.2350	.0602	.0630	.0646
7	.4002	.2267	.04036	1614	.2502	.2350	.0603	.0633	.0654
8	.4007	.2262	.04018	1607	.2493	.2350	.0593	.0630	.0657
9	.4004	.2266	.04033	1613	.2494	.2350	.0592	.0624	.0644
10	.4001	.2261	.04015	1606	.2486	.2350	.0587	.0618	.0635
Average	.....	.....	.....	.....	.....	.....	.0592	.0626	.0646

No.	$L-L_4$	D_4 maximum	D_4 minimum	$L-L_5$	D_5 maximum	D_5 minimum	$L-L_6$	$L-L_7$	$L-L_8$
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch
1	.....	0.2525	0.2386	.....	0.2530	0.2405	.....	.....	.....
2	0.0673	.2514	.2386	0.0687	.2517	.2405	0.0697	0.0709	0.0735
3	.0682	.2514	.2386	.0696	.2522	.2405	.0714	.0736	.0753
4	.....	.2498	.2386	.....	.2505	.2405	.....	.....	.....
5	.0659	.2505	.2386	.0670	.2510	.2405	.0682	.0694	.0703
6	.0670	.2519	.2386	.0702	.2532	.2405	.0708	.0719	.0726
7	.0666	.2520	.2386	.0682	.2534	.2405	.0698	.0721	.0740
8	.0685	.2516	.2386	.0700	.2526	.2405	.0709	.0727	.0746
9	.0675	.2519	.2386	.0693	.2534	.2405	.0717	.0734	.0748
10	.0665	.2508	.2386	.0675	.2518	.2405	.0693	.0710	.0732
Average	.0672	.....	.....	.0688	.....	.....	.0702	.0718	.0735

TABLE 4.—Experiments on Repeated Application of Same Load

No.	L	D	A	Load equivalent to 32 000 lbs./in.²	D_1 maximum	D_1 minimum	$L-L_1$
	Inch	Inch	Inch²	Pounds	Inch	Inch	Inch
1	0.4001	0.2260	0.04012	1284	0.2408	0.2295	0.0410
2	.3998	.2260	.04012	1284	.2408	.2295	.0389
3	.3999	.2264	.04026	1288	.2409	.2295	.0388
4	.4000	.2270	.04047	1295	.2425	.2295	.0409
5	.4002	.2264	.04026	1288	.2413	.2295	.0401
6	.3998	.2264	.04026	1288	.2410	.2295	.0391
7	.4005	.2262	.04018	1286	.2410	.2295	.0400
8	.4002	.2264	.04026	1288	.2414	.2295	.0402
9	.4000	.2264	.04026	1288	.2414	.2295	.0404
10	.3997	.2260	.04012	1284	.2415	.2295	0403
Average	.....	.....	.....	.....	.....	.....	.0400

TABLE 4.—Experiments on Repeated Application of Same Load—Continued

No.	$L-L_2$	$L-L_3$	$L-L_4$	$L-L_5$	D_6 maximum	D_6 minimum	$L-L_6$
	Inch	Inch	Inch	Inch	Inch	Inch	Inch
1	0.0425	0.0450	0.0481	0.0492	0.2442	0.2345	0.0510
2	.0407	.0430	.0445	.0454	.2430	.2345	.0472
3	.0407	.0438	.0458	.0469	.2438	.2345	.0488
4	.0430	.0444	.0458	.0473	.2447	.2345	.0480
5	.0420	.0434	.0445	.0460	.2435	.2345	.0474
6	.0418	.0445	.0455	.0469	.2435	.2345	.0477
7	.0417	.0432	.0446	.0472	.2438	.2345	.0489
8	.0426	.0441	.0461	.0482	.2440	.2345	.0494
9	.0430	0456	.0468	.0482	.2430	.2345	.0497
10	.0423	.0450	.0469	.0480	.2442	.2345	.0498
Average	.0420	.0442	.0459	.0473			.0488

TABLE 5.—Experiments on Repeated Application of Same Load

No.	L	D	A	Load equivalent to 20 000 lbs./in.^{2}	D_1 maximum	D_1 minimum	$L-L_1$
	Inch	Inch	Inch^2	Pounds	Inch	Inch	Inch
1	0.4000	0.2256	0.03997	799	0.2315	0.2265	0.0165
2	.4001	.2257	.04001	800	.2319	.2265	.0161
3	.4000	.2267	.04036	807	.2332	.2270	.0172
4	.4000	.2257	.04001	800	.2328	.2260	.0185
5	.4002	.2263	.04022	804	.2332	.2265	0177
6	.4000	.2263	.04022	804	.2323	.2270	.0161
7	.3999	.2262	.04019	803	.2328	.2265	.0172
8	.4000	.2255	.03994	799	.2323	.2265	.0175
9	.4000	.2261	.04015	803	.2319	.2265	.0156
10	.4002	.2263	.04022	804	.2324	.2265	.0164
Average							.0169

No.	$L-L_2$	$L-L_3$	$L-L_4$	$L-L_5$	D_6 maximum	D_6 minimum	$L-L_6$
	Inch	Inch	Inch	Inch	Inch	Inch	Inch
1	0.0187	0.0196	0.0202	0.0207	0.2328	0.2264	0.0216
2	.0181	.0194	.0207	.0208	.2334	.2264	.0220
3	.0182	.0194	.0205	.0213	.2343	.2264	.0218
4	.0196						
5	.0199	.0213	.0221	.0233	.2352	.2264	.0246
6	.0173	.0182	.0191	.0197	.2336	.2264	.0209
7	.0191	.0204	.0212	.0228	.2346	.2264	.0235
8	.0195	.0207	.0219	.0223	.2338	.2264	.0236
9	.0164	.0184	.0191	.0200	2341	.2264	.0205
10	.0175	.0188	.0205	.0214	.2340	.2264	.0226
Average	.0184	.0196	.0206	.0214			.0223

NOTES ON TABLES 3, 4, AND 5.—These tables represent the results of experiments with cylinders of Series No. 2 by subjecting them to repeated compressions with single application of same load for about 2½ seconds.

The repeated loads were:

Table	Load
	Lbs./in.²
3.....	40 000
4.....	32 000
5.....	20 000

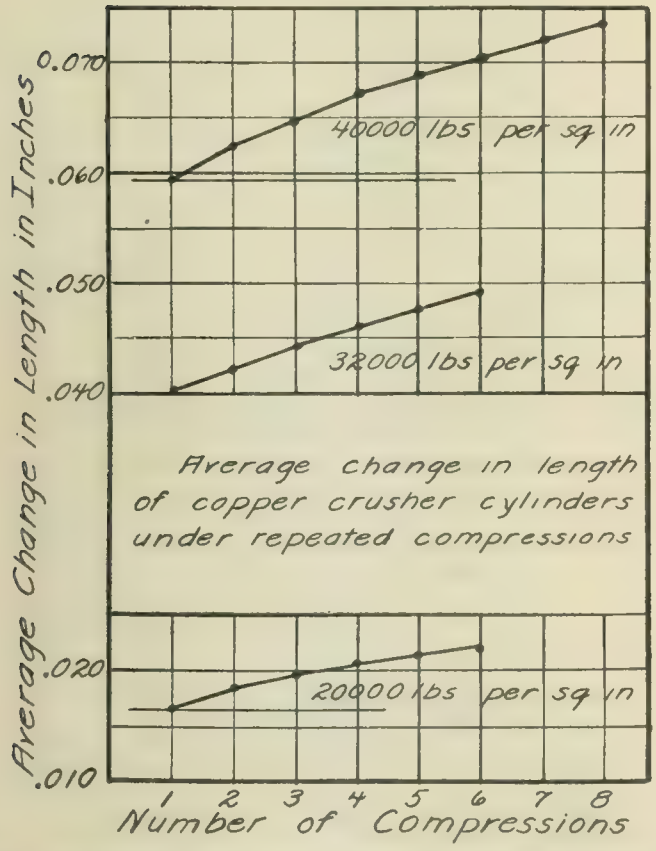

FIG. 5.—*Change of length of copper cylinders upon repeated compression*

All results are shown in Fig. 5.

The following conclusions may be drawn from these curves:

(1) The length of cylinders decreases considerably with the number of times the load is applied.

(2) This change also decreases slightly with number of application of load as the hardness of copper increases.

(3) The slope of the curve representing results of repeated compression with the load P is steeper than slope of curve representing results of repeated compression with the load Q if load P is greater than Q.

(*b*) EXPERIMENTS ON EFFECT OF DURATION OF COMPRESSION APPLIED UNDER CONDITION (1)

TABLE 6.—Experiments on Effect of Duration of Compression Applied Under Condition (1)

L_1 = length of cylinders after applying pressure of 40 000 lbs./in.^{2} for about $2\frac{1}{2}$ seconds.

No.	L	D	A	Load equivalent to 40 000 lbs./in.^{2}	$L-L_1$	No.	L	D	A	Load equivalent to 40 000 lbs./in.^{2}	$L-L_1$
	Inch	Inch	Inch^2	Lbs.	Inch		Inch	Inch	Inch^2	Lbs.	Inch
1	0.3999	0.2258	0.04004	1601	0.0587	10	0.3999	0.2264	0.04026	1610	0.0583
2	.4001	.2262	.04018	1607	.0588	11	.4002	.2260	.04012	1604	.0589
3	.4002	.2266	.04033	1613	.0587	12	.4001	.2257	.04001	1600	.0579
4	.4000	.2257	.04001	1600	.0570	13	.3999	.2259	.04008	1603	.0609
5	.3997	.2263	.04022	1609	.....	14	.4000	.2264	.04026	1610	.0596
6	.3999	.2260	.04012	1604	.0579	15	.3999	.2257	.04001	1600	.0592
7	.3998	.2265	.04029	1612	.0591	Average	.....	.....	.....	.....	.0588
8	.4001	.2259	.04008	1603	.0589						
9	.3998	.2260	.04012	1604	.0601						

TABLE 7.—Experiments on the Effect of Duration of Compression Applied Under Condition (1)

L_1 = length of cylinders after applying pressure of 40 000 lbs./in.^{2} for about 25 seconds.

No.	L	D	A	Load equivalent to 40 000 lbs./in.^{2}	$L-L_1$
	Inch	Inch	Inch^2	Pounds	Inch
1	0.3999	0.2256	0.03997	1598	0.0601
2	.4002	.2264	.04026	1610	.....
3	.3999	.2262	.04018	1607	.....
4	.4002	.2265	.04029	1612	.0594
5	.3999	.2262	.04018	1607	.0609
6	.3999	.2261	.04015	1606	.0600
7	.3998	.2256	.03997	1598	.0604
8	.4001	.2263	.04022	1609	.0603.
9	.3999	.2267	.04036	1614	.0600
10	.4000	.2262	.04018	1607	.0579
11	.4002	.2257	.04001	1600	.0580
12	.4002	.2257	.04001	1600	.0608
13	.4000	.2264	.04026	1610	.0578
14	.3998	.2262	.04018	1607	.....
15	.3999	.2263	.04022	1609	.0602
16	.3998	.2261	.04015	1606	.0611
Average	.....	.....	.....	.....	.0598

TABLE 8.—Experiments on Effect of Duration of Compression Applied Under Condition (1)

L_1=length of cylinders after applying pressure of 40 000 lbs./in.² for about 5 seconds.

No.	L	D	A	Load equivalent to 40 000 lbs./in.²	$L-L_1$
	Inch	Inch	Inch²	Pounds	Inch
1	0.3999	0.2266	0.04033	1613	0.0579
2	.4009	.2263	.04022	1609	.0600
3	.4002	.2256	.03997	1598	.0597
4	.4001	.2265	.04029	1612	.0593
5	.4001	.2260	.04012	1604	.0585
6	.3998	.2258	.04004	1601	.0583
7	.4002	.2256	.03997	1598	.0592
8	.4002	.2265	.04029	1612	.0592
9	.3999	.2265	.04029	1612	.0595
10	.4000	.2256	.03997	1598	.0586
11	.4000	.2257	.04001	1600	.0593
12	.4000	.2267	.04036	1614	.....
13	.4002	.2257	.04001	1600	.0609
14	.3998	.2260	.04012	1604	.0595
15	.3999	.2259	.04008	1603	.0609
Average	.....	.....	.....	.....	.0593

NOTES ON TABLES 6, 7, AND 8.—In this case experiments were made upon the cylinders of Series No. 2. Experiments consisted of compressions of 40 000 pounds per square inch applied under condition (1) for different periods, 2½, 5, and 25 seconds. Average changes in length are 0.0588, 0.0593, and 0.0598 inch, respectively. These experiments show slight increase of total set with increased time of application of load.

(*c*) EXPERIMENTS ON EFFECT OF TEMPERATURE OF ANNEALING

TABLE 9.—Experiments on Effect of Temperature of Annealing

[These copper cylinders were annealed at 1200° F (650° C).]

No.	L	D	A	Load equivalent to 30 000 lbs./in.²	D_1 maximum	$L-L_1$	Load equivalent to 35 000 lbs./in.²	$L-L_2$	Load equivalent to 40 000 lbs./in.²	$L-L_3$
	Inch	Inch	Inch²	Pounds	Inch	Inch	Pounds	Inch	Pounds	Inch
1	0.5002	0.2223	0.03881	1164	0.2355	0.0478	1358	0.0628	1552	0.0774
2	.5000	.2225	.03888	1166	.2356	.0478	1361	.0626	1555	.0780
3	.5000	.2226	.03892	1168	.2359	.0482	1362	.0624	1557	.0775
4	.5000	.2215	.03853	1156	.2342	.0473	1349	.0614	1541	.0764
5	.5003	.2221	.03874	1162	.2351	.0476	1356	.0624	1550	.0772
6	.5000	.2226	.03892	1168	.2357	.0487	1362	.0626	1557	.0783
7	.5002	.2231	.03909	1173	.2360	.0472	1368	.0610	1564	.0772
8	.5001	.2228	.03899	1170	.2361	.0480	1365	.0627	1560	.0774
9	.5001	.2225	.03888	1166	.2359	.0489	1361	.0627	1555	.0786
10	.4999	.2225	.03888	1166	.2355	.0477	1361	.0621	1555	.0772
11	.5002	.2218	.03864	1159	.2345	.0476	1352	.0616	1546	.0769
12	.5000	.2222	.03878	1163	.2353	.0486	1357	.0630	1551	.0783
Average	.....	.....	.....	.....	.....	.0480	.....	.0623	.....	.0775

TABLE 10.—Experiments on Effect of Temperature of Annealing

[These cylinders were annealed at 1000° F (540° C)]

No.	L	D	A	Load equivalent to 30 000 lbs./in.^{2}	$L-L_1$	D_1 maximum	Load equivalent to 35 000 lbs./in.^{2}	$L-L_2$	Load equivalent to 40 000 lbs./in.^{2}	$L-L_3$
	Inch	Inch	Inch^2	Pounds	Inch	Inch	Pounds	Inch	Pounds	Inch
1	0.5003	0.2249	0.03973	1192	0.0456	0.2382	1391	0.0597	1589	0.0749
2	.5005	.2248	.03969	1191	.0465	.2378	1389	.0603	1588	.0756
3	.5010	.2255	.03994	1198	.0446	.2378	1398	.0589	1598	.0737
4	.5007	.2250	.03976	1193	.0464	.2381	1392	.0603	1590	.0756
5	.5007	.2257	.04001	1200	.0466	.2386	1400	.0609	1600	.0765
6	.5004	.2252	.03983	1195	.0458	.2382	1394	.0609	1593	.0748
7	.5004	.2250	.03976	1193	.0465	.3281	1392	.0606	1590	.0763
8	.5003	.2253	.03987	1196	.0450	.2379	1395	.0595	1595	.0733
9	.5004	.2253	.03987	1196	.0458	.2379	1395	.0601	1595	.0746
10	.5002	.2251	.03980	1194	.0461	.2378	1393	.0599	1592	.0748
11	.5005	.2254	.03990	1197	.0460	.2382	1397	.0604	1596	.0751
12	.5002	.2254	.03990	1197	.0454	.2379	1397	.0593	1596	.0744
Average					.0459			.0601		.0750

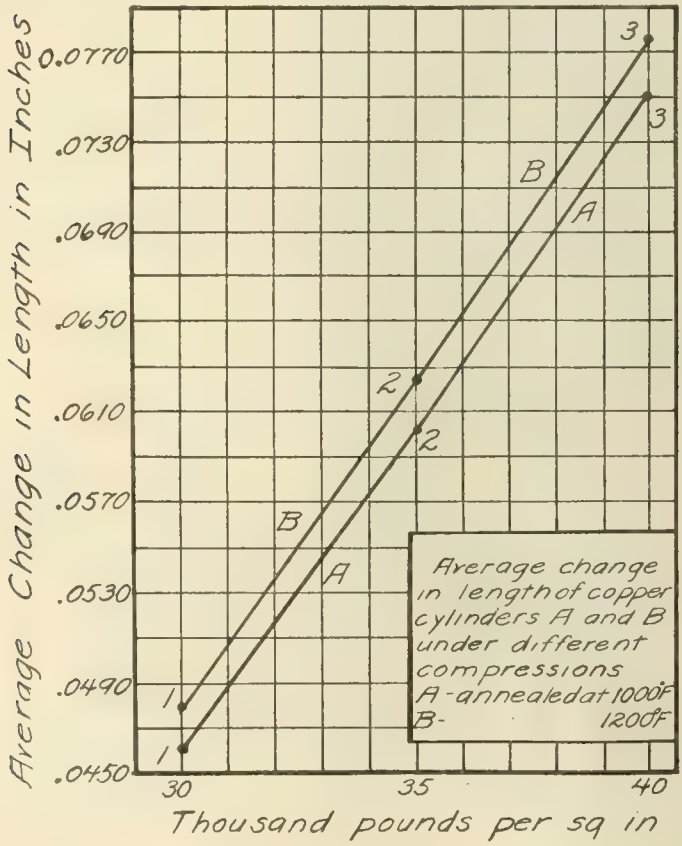

FIG. 6.—*Effect of annealing upon the change of length of copper cylinders upon compression*

NOTES ON TABLES 9 AND 10.—These tables represent experiments with copper cylinders of Series No. 3, annealed at 1000 and 1200° F (540 and 650° C). Different compressions (30 000, 35 000, and 40 000 pounds per square inch) were made keeping the load under condition (1) for about 2½ seconds. Fig. 6 indicates that copper cylinders annealed at 1200° F are softer than those annealed at 1000° F.

(*d*) EXPERIMENTS ON INFLUENCE OF INITIAL PRECOMPRESSION WHEN LOADS SUCCESSIVELY APPLIED DIFFER CONSIDERABLY FROM EACH OTHER

TABLE 11.—Experiments on Influence of Initial Precompression when Loads Successively Applied Differ Considerably from Each Other

L_1=length of cylinders after applying pressure of 20 000 lbs./in.² for about 2½ seconds.
L_2=length of cylinders after applying pressure of 32 000 lbs./in.² for about 2½ seconds.
L_3=length of cylinders after applying pressure of 40 000 lbs./in.² for about 2½ seconds.

No.	L	D	A	Load equivalent to 20 000 lbs./in.²	$L-L_1$	D_1 maximum	Load equivalent to 32 000 lbs./in.²	$L-L_2$	D_2 maximum	Load equivalent to 40 000 lbs./in.²	$L-L_3$	D_3 maximum
	Inch	Inch	Inch²	Lbs.	Inch	Inch	Lbs.	Inch	Inch	Lbs.	Inch	Inch
1	0.4000	0.2260	0.04012	802	0.0161	0.2322	1284	0.0395	0.2410	1605	0.0585	0.2481
2	.3998	.2264	.04026	805	.0168	.2331	1288	.0404	.2418	1610	.0596	.2494
3	.3998	.2257	.04001	800	.0175	.2327	1280	.0410	.2413	1600	.0607	.2494
4	.4000	.2259	.04008	802	.0180	.2323	1283	.0402	.2403	1603	.0595	.2487
5	.4000	.2253	.03987	797	.0156	.2318	1276	.0400	.2403	1595	.....	.....
6	.3998	.2261	.04015	803	.0154	.2322	1285	.0398	.2408	1606	.0594	.2490
7	.3998	.2260	.04012	802	.0158	.2317	1284	.0400	.2402	1605	.0601	.2486
8	.4000	.2260	.04012	802	.0150	.2317	1284	.0375	.2400	1605	.0560	.2476
9	.3999	.2263	.04022	804	.0160	.2322	1287	.0399	.2414	1609	.0586	.2488
10	.4001	.2263	.04022	804	.0164	.2327	1287	.0399	.2412	1609	.0589	.2490
11	.3998	.2265	.04029	806	.0175	.2333	1289	.0413	.2427	1612	.0603	.2498
12	.4000	.2265	.04029	806	.0167	.2326	1289	.0396	.2412	1612	.0588	.2488
13	.4000	.2258	.04004	801	.0171	.2322	1281	.0407	.2413	1602	.0607	.2494
14	.3999	.2263	.04022	804	.0176	.2331	1287	.0406	.2415	1609	.0598	.2493
15	.3998	.2268	.04040	808	.0158	.2332	1293	.0396	.2420	1616	.0590	.2498
Average	.....	.....	.....	.....	.0165	.....	.....	.0400	.....	.....	.0593	.....

NOTES ON TABLE 11.—Here are represented experiments with copper cylinders of Series No. 2. Successive compressions (20 000, 32 000, and 40 000 pounds per square inch) were made on same cylinders, keeping the load under condition (1) for about 2½ seconds.

Comparing the average changes in length of cylinders with those of other tables after the first compression, we can conclude that in this case change in length depends only on value of load of the last compression, no matter whether previous compressions had been made or not.

After the first compression of 32 000 pounds per square inch, Tables 4 and 12 show an average $L-L_1$ of 0.0400 inch. After two compressions, the first of which was 20 000 pounds per square inch and the last 32 000 pounds per square inch, Table No. 11 shows an average $L-L_2$ of 0.0400 inch.

After first compression of 40 000 pounds per square inch:

Table No.	Results of first compression $L-L_1$
	Inch
6	0.0588
3	.0592
16	.0589
19	.0588

After three compressions, the first of which was 20 000 pounds per square inch, the second 32 000, and the third 40 000 pounds per square inch, Table 11 shows an average $L-L_3$ of 0.0593 inch. Comparing these figures—that is, 0.0400 with 0.0400 inch, and 0.0593 with 0.0588, 0.0592, 0.0589, and 0.0588 inch—we see that a very close concordance was obtained. It must be noted, however, that this statement may apply only when previous loads differ considerably from the last load, as was the case in these experiments. (The first load was 20 000, the second load 32 000, and the last load was 40 000 pounds per square inch.)

(*e*) EXPERIMENTS ON EFFECT OF INITIAL PRECOMPRESSION WHEN DIFFERENCES BETWEEN LOADS SUCCESSIVELY APPLIED ARE ONLY SLIGHT

TABLE 12.—Experiments on the Effect of Initial Precompression when the Differences Between the Loads Successively Applied are only Slight

L_1=length of cylinders after pressure of 32 000 lbs./in.^{2} for about 2½ seconds.
L_2=length of cylinders after pressure of 36 000 lbs./in.^{2} for about 2½ seconds.
L_3=length of cylinders after pressure of 38 000 lbs./in.^{2} for about 2½ seconds.
L_4=length of cylinders after pressure of 40 000 lbs./in.^{2} for about 2½ seconds.

No.	L	D	A	Load equivalent to 32 000 lbs./in.^{2}	$L-L_1$	D_1 maximum	Load equivalent to 36 000 lbs./in.^{2}
	Inch	Inch	Inch^2	Pounds	Inch	Inch	Pounds
1	0.3998	0.2258	0.04004	1281			
2	.3999	.2262	.04018	1286	0.0403	0.2410	1446
3	.3999	.2257	.04001	1280	.0405	.2410	1440
4	.3999	.2261	.04015	1285	.0384	.2411	1445
5	.3997	.2258	.04004	1281	.0386	.2407	1441
6	.3997	.2267	.04036	1292	.0407	.2425	1453
7	.3999	.2267	.04036	1292	.0409	.2422	1453
8	.3999	.2263	.04022	1287	.0391	.2415	1448
9	.3999	.2265	.04029	1289	.0392	.2417	1450
10	.3999	.2267	.04036	1292	.0409	.2423	1453
11	.3998	.2267	.04036	1292	.0408	.2425	1453
12	.3997	.2260	.04012	1284	.0399	.2413	1444
13	.3995	.2259	.04008	1283	.0398	.2410	1443
14	.3996	.2263	.04022	1287	.0408	.2412	1448
15	.3997	.2268	.04040	1293			
Average					.0400		

TABLE 12.—Experiments on the Effect of Initial Precompression when the Differences Between the Loads Successively Applied are only Slight—Continued

No.	$L-L_2$	Load equivalent to 38 000 lbs./in.²	$L-L_3$	Load equivalent to 40 000 lbs./in.²	$L-L_4$	Load D_4 maximum
	Inch	Pounds	Inch	Pounds	Inch	Inch
1						
2	0.0496	1527	0.0570	1607	0.0624	0.2486
3	.0502	1520	.0561	1600	.0624	.2495
4	.0496	1526	.0553	1606	.0612	.2493
5	.0495	1522	.0550	1602	.0607	.2491
6	.0502	1534	.0573	1614	.0627	.2496
7	.0497	1534	.0557	1614	.0618	.2498
8	.0499	1528	.0560	1609	.0616	.2494
9	.0505	1531	.0563	1612	.0624	.2494
10	.0506	1534	.0568	1614	.0629	.2500
11	.0509	1534	.0575	1614	.0638	.2508
12	.0495	1525	.0551	1605	.0618	.2488
13	.0497	1523	.0553	1603	.0609	.2489
14	.0504	1528	.0566	1609	.0620	.2494
15						
Average	.0500		.0562		.0620	

TABLE 13.—Experiments on Effect of Initial Precompression when Differences Between Loads Successively Applied are only Slight

L_1=length of cylinders after applying pressure of 36 000 lbs./in.² for about 2½ seconds.
L_2=length of cylinders after applying pressure of 38 000 lbs./in.² for about 2½ seconds.
L_3=length of cylinders after applying pressure of 40 000 lbs./in.² for about 2½ seconds.

No.	L	D	A	Load equivalent to 36 000 lbs./in.²	$L-L_1$	D_1 maximum	Load equivalent to 38 000 lbs./in.²	$L-L_2$	Load equivalent to 40 000 lbs./in.²	$L-L_3$	D_3 maximum
	Inch	Inch	Inch²	Pounds	Inch	Inch	Pounds	Inch	Pounds	Inch	Inch
1	0.3995	0.2260	0.04012	1444	0.0488	0.2447	1525	0.0545	1605	0.0610	0.2489
2	.3995	.2260	.04012	1444	.0486	.2445	1525	.0550	1605	.0601	.2488
3	.3994	.2263	.04022	1448	.0483	.2448	1528	.0549	1609	.0607	.2490
4	.3997	.2268	.04040	1454	.0500	.2462	1535	.0559	1616	.0622	.2506
5	.3997	.2262	.04018	1446	.0488	.2448	1527	.0557	1607	.0615	.2492
6	.3998	.2266	.04033	1452	.0501	.2451	1533	.0562	1613		
7	.3994	.2257	.04001	1440	.0483	.2442	1520	.0553	1600	.0609	.2489
8	.3994	.2267	.04036	1453	.0496	.2458	1534	.0561.	1614	.0619	.2501
9	.3998	.2254	.03990	1436	.2447	.2448	1516	.0555	1596	.0613	.2486
10	.3997	.2255	.03994	1438	.0500	.2452	1518	.0564	1598		
11	.4000	.2265	.04029	1450	.0499	.2455	1531	.0565	1612	.0625	.2500
12	.3996	.2258	.04004	1441	.0480	.2445	1522	.0550	1602	.0604	.2486
13	.4000	.2257	.04001	1440	.0505	.2451	1520	.0563	1600	.0628	.2492
14	.3999	.2256	.03997	1439	.0485	.2440	1519	.0548	1599	.0606	.2483
15	.3998	.2258	.04004	1441	.0501	.2450	1522	.0563	1602	.0625	.2495
Average					.0493			.0556		.0614	

TABLE 14.—Experiments on the Effect of Initial Precompression when the Differences Between the Loads Successively Applied are only Slight

L_1=length of cylinders after applying pressure of 38 000 lbs./in.^{2} for about 2½ seconds.
L_2=length of cylinders after applying pressure of 40 000 lbs./in.^{2} for about 2½ seconds.

No.	L	D	A	Load equivalent to 38 000 lbs./in.^{2}	$L-L_1$	D_1 maximum	Load equivalent to 40 000 lbs./in.^{2}	$L-L_2$	D_2 maximum
	Inch	Inch	Inch^3	Lbs.	Inch	Inch	Lbs.	Inch	Inch
1	0.3998	0.2267	0.04036	1534	0.0550	0.3483	1614	0.0620	0.2503
2	.4000	.2260	.04012	1525	.0528	.3465	1605	.....	.....
3	.4001	.2262	.04018	1527	.0533	.3466	1607	.0597	.2487
4	.3999	.2255	.03994	1518	.0549	.3466	1598	.0616	.2490
5	.3999	.2256	.03997	1519	.0530	.3465	1599	.0595	.2489
6	.4000	.2262	.04018	1527	.0540	.3467	1607	.0609	.2494
7	.3998	.2257	.04001	1520	.0556	.3468	1600	.0621	.2492
8	.4001	.2262	.04018	1527	.0540	.3464	1607	.0604	.2489
9	.4001	.2259	.04008	1523	.0528	.3467	1603	.0599	.2489
10	.4002	.2264	.04026	1530	.0538	.3470	1610	.0598	.2496
11	.4000	.2257	.04001	1520	.0546	.3469	1600	.0608	.2486
12	.4000	.2264	.04026	1530	.0540	.3468	1610	.0607	.2496
13	.4001	.2256	.03997	1519	.0539	.3463	1599	.0597	.2486
14	.4002	.2260	.04012	1525	.....	.....	.....	.....	.....
15	.4003	.2263	.04022	1528	.0549	.3470	1609	.0608	.2492
Average	.....	.....	.....	.....	.0540	.....	.....	.0606	.....

TABLE 15.—Effect of Initial Precompression when Differences Between Loads Successively Applied are only Slight

Reference	32 000 lbs./in.^{2}	36 000 lbs./in.^{2}	38 000 lbs./in.^{2}	40 000 lbs./in.^{2}
See Tables Nos. 4, 11, and 12	$L-L_1=0.0400$	$L-L_2=0.0500$	$L-L_3=0.0562$	$L-L_4=0.0620$
See Table No. 13	.....	$L-L_1=0.0493$	$L-L_2=0.0556$	$L-L_3=0.0614$
See Table No. 14	.....	.....	$L-L_1=0.0540$	$L-L_2=0.0606$
See Tables Nos. 3, 6, 11, 16, and 19	.....	.....	.....	$L-L_1=0.0590$

NOTES ON TABLES 12, 13, 14, AND 15.—Experiments were made upon cylinders of the same stock; that is, cylinders of the Series No. 2.

Successive compressions of 32 000, 36 000, 38 000, and 40 000 pounds per square inch were made under condition (1), keeping the load on for about 2½ seconds.

Average changes in length under initial and successive compressions are shown in Table 15. Here are collected from Tables 3, 4, 6, 11, 12, 13, 14, 16, and 19 average changes of length (total sets). These data show the difference between the change in length of cylinders which were compressed at load P and of those previously compressed using a load Q slightly lower than load P and then compressed at load P.

It should be noted that in the case where several successive loads, differing by 2000 to 4000 pounds per square inch, had previously been applied the total change of length increases as the number of loads previously applied increases. All these results agree with experiments made with repeated compressions of same and smaller (within certain limits) loads applied on copper which is known to have practically no yield point.

(*f*) EXPERIMENTS ON EFFECT OF DURATION OF COMPRESSION APPLIED UNDER CONDITION (2) AND COMPARISON OF RESULTS OF DIFFERENT METHODS OF LOADING

TABLE 16.—Experiments on Effect of Duration of Compression Applied Under Condition (2)

L_1=length of cylinders after applying load of 40 000 lbs./in.^{2} for about 2½ seconds.

No.	L	D	A	Load equivalent to 40 000 lbs./in.^{2}	$L-L_1$	D_1 maximum
	Inch	Inch	Inch ^{2}	Pounds	Inch	Inch
1	0.4000	0.2261	0.04015	1606	0.0582	0.2485
2	.4004	.2259	.04008	1603	.0598	.2488
3	.4000	.2259	.04008	1603	.0587	.2487
4						
5	.4003	.2259	.04008	1603	.0586	.2482
6	.3997	.2262	.04018	1607	.0579	.2484
7	.4005	.2265	.04029	1612	.0588	.2486
8	.4000	.2260	.04012	1604	.0588	.2488
9	.4000	.2266	.04033	1613	.0594	.2496
10	.4001	.2256	.03997	1599	.0601	.2489
Average					.0589	

TABLE 17.—Experiments on Effect of Duration of Compression Applied Under Condition (2)

L_1=length of cylinders after applying pressure of 40 000 lbs./in.^{2}, holding beam balanced for about 5 seconds.

No.	L	D	A	Load equivalent to 40 000 lbs./in.^{2}	$L-L_1$	D_1 maximum
	Inch	Inch	Inch ^{2}	Pounds	Inch	Inch
1	0.3996	0.2263	0.04022	1609		0.2494
2	.3996	.2266	.04033	1613	.0589	.2497
3	.3995	.2263	.04022	1609	.0598	.2494
4	.3996	.2257	.04001	1600	.0611	.2494
5	.3995	.2268	.04040	1616	.0596	.2505
6	.3995	.2265	.04029	1612	.0594	.2495
7	.3996	.2259	.04008	1603		.2475
8	.3996	.2253	.03987	1595	.0596	.2490
9	.3996	.2263	.04022	1609		.2503
10	.3995	.2263	.04022	1609	.0590	.2493
11	.4002	.2258	.04004	1602	.0604	.2486
12	.4000	.2255	.03994	1598	.0594	.2487
Average					.0597	

TABLE 18.—Experiments on Effect of Duration of Compression Applied Under Condition (2)

L_1 = length of cylinders after applying pressure of 40 000 lbs./in.,² holding beam balanced for about 25 seconds.

	L	D	A	Load equivalent to 40 000 lbs./in.²	$L-L_1$	D_1 maximum
	Inch	Inch	Inch²	Pounds	Inch	Inch
1	0.4007	0.2262	0.04018	1607	0.0615	0.2495
2	.4003	.2261	.04015	1606	.0591	.2489
3	.4005	.2257	.04001	1600	.0590	.2485
4	.4003	.2262	.04018	1607	.0608	.2494
5	.4006	.2266	.04033	1613	.0609	.2505
6	.4006	.2257	.04001	1600	.0604	.2494
7	.4005	.2256	.03997	1599	.0613	.2488
8	.4005	.2258	.04004	1602	.0615	.2495
9	.4004	.2255	.03994	1598	.0611	.2486
10	.4008	.2263	.04022	1609	.0590	.2487
11	.4008	.2265	.04029	1612	.0611	.2495
12	.4008	.2264	.04026	1610	.0601	.2492
Average					.0605	

NOTES ON TABLES 16, 17, AND 18.—These tables represent the results of experiments made with cylinders of same series, No. 2, by subjecting them to compressions of 40 000 pounds per square inch applied under condition (2) for different periods—2½, 5, and 25 seconds—that is, holding beam balanced for these different periods.

In order to keep the beam balanced in this case only one additional application of same load (impulse) was needed after the beam had started to drop. Average changes in length are 0.0589, 0.0597, and 0.0605 inch, respectively.

It may be of interest to compare these results with those obtained in the experiments where the same load, 40 000 pounds per square inch, was held for different periods after the balance had been obtained and the beam allowed to drop, and also with those results obtained where the same load of 40 000 pounds per square inch was applied successively two times (for 2½ seconds each time), the second load being applied at some interval after the first load was entirely released.

1. In the case when load of 40 000 pounds per square inch was held for different periods after balance was obtained, the beam being allowed to drop (Tables 6, 7, and 8), the decrease in length of cylinders as compared with the length after compression for

2½ seconds (considering average change for 2½ seconds is equal to 0.0590 inch) is shown here:

Time of compression	Difference in decrease
Seconds	Inch
2½	0.0000
5	.0003
25	.0008

2. In case where same load of 40 000 pounds per square inch was held for different periods after balance was obtained, but holding beam balanced by means of one additional application of same load (Tables 16, 17, and 18), these differences were:

Time of compression	Difference in decrease
Seconds	Inch
2½	0.0000
5	.0007
25	.0015

3. In the case of one repeated application of the same load of 40 000 pounds per square inch (Table 4) the difference between the decrease in length after first and second applications for 2½ seconds each is 0.0626—0.0590 inch = 0.0036 inch. Hence it may be seen: (*a*) That the application of the same load for the same period (greater than 3 seconds), but made under condition (2)—that is, holding the beam balanced by means of one additional application (impulse) of the same load—causes about twice the decrease in length observed in the case when the beam is allowed to drop after the balance is obtained; (*b*) that where the load is applied twice (for 2½ seconds each time), the second load being applied at some intervals after the first load has been entirely released, the decrease in length is much greater than in the first two cases, even when the load is held there as long as 25 seconds.

(*g*) EXPERIMENTS ON AGING OF COPPER CYLINDERS OF SERIES No. 2, TABLES 19-25

The duration of the compression in these experiments was about 2½ seconds.

TABLE 19.—Compression Tests Relating to Experiments on Aging of Copper Cylinders

L_1 = length of cylinders after applying pressure of 40 000 lbs./in.²

No.	L	D	A	Load equivalent to 40 000 lbs./in.²	$L-L_1$
	Inch	Inch	Inch²	Pounds	Inch
1	0.4001	0.2266	0.04033	1613	0.0599
2	.4002	.2259	.04008	1603	.0587
3	.4001	.2265	.04029	1612	.0596
4	.4001	.2258	.04004	1602	.0611
5	.4000	.2258	.04004	1602	.0592
6	.4001	.2259	.04008	1603	.0558
7	.4000	.2253	.03987	1595	.0618
8	.4000	.2255	.03994	1598	.0555
9	.4002	.2260	.04012	1605	.0592
10	.4003	.2265	.04029	1612	.0606
11	.4001	.2262	.04018	1607	.0578
12	.4002	.2264	.04026	1610	.0551
13	.4001	.2257	.04001	1600	.0608
14	.4004	.2258	.04004	1602	.0598
15	.4003	.2266	.04033	1613	.0579
Average					.0588

NOTES ON TABLE 19.—The table shows the mean total set of cylinders compressed at 40 000 pounds per square inch. The total set was 0.0599 inch.

TABLE 20.—Compression Tests Relating to Experiments on Aging of Copper Cylinders

L_1 = length of cylinders after applying a pressure of 38 000 lbs./in.² for about 2½ seconds.
L_2 = length of cylinders after applying a pressure of 40 000 lbs./in.² for about 2½ seconds, the pressure of 40 000 lbs./in.² being applied 1 hour after the pressure of 38 000 lbs.

No.	L	D	A	Load equivalent to 38 000 lbs./in.²	$L-L_1$	Load equivalent to 40 000 lbs./in.²	$L-L_2$
	Inch	Inch	Inch²	Pounds	Inch	Pounds	Inch
1	0.4002	0.2260	0.04012	1525	0.0530	1605	0.0612
2	.4002	.2261	.04015	1526	.0552	1606	.0622
3	.4004	.2257	.04001	1520	.0556	1600	.0619
4	.4001	.2265	.04029	1531	.0541	1612	.0616
5	.4001	.2260	.04012	1525	.0554	1605	.0616
6	.4005	.2262	.04018	1527	.0539	1607	.0594
7	.4001	.2258	.04004	1522	.0551	1602	.0608
8	.4003	.2263	.04022	1528	.0529	1609	.0595
9	.4005	.2263	.04022	1528	.0547	1609	.0612
10	.4001	.2259	.04008	1523	.0514	1603	.....
11	.4002	.2261	.04015	1526	.0517	1606	.0588
12	.4001	.2257	.04001	1520	.0545	1600	.0616
13	.4003	.2258	.04004	1522	.0513	1602	.....
14	.4003	.2258	.04004	1522	.0544	1602	.0606
15	.4005	.2257	.04001	1520	.0543	1600	.0607
Average					.0538		.0608

NOTES ON TABLE 20.—The cylinders were precompressed at 38 000 pounds per square inch, and one hour later they were compressed at 40 000 pounds per square inch. Total set was 0.0608 inch.

TABLE 21.—Compression Tests Relating to Experiments on Aging of Copper Cylinders

L_1 = length of cylinders after applying a pressure of 38 000 lbs./in.^{2} for about 2½ seconds
L_2 = length of cylinders after applying a pressure of 40 000 lbs./in.^{2} for about 2½ seconds, the pressure of 40 000 lbs. being applied after aging cylinders at 100° C during 28 days.

No.	L	D	A	Load equivalent to 38 000 lbs./in.^{2}	$L-L_1$	Load equivalent to 40 000 lbs./in.^{2}	$L-L_2$
	Inch	Inch	Inch^2	Pounds	Inch	Pounds	Inch
1	0.4005	0.2257	0.04001	1520	0.0551	1600	0.0632
2	.4003	.2258	.04004	1522	.0544	1602	.0626
3	.4006	.2257	.04001	1520	.0566	1600	.0627
4	.4004	.2257	.04001	1520	.0519	1600	.0596
5	.4005	.2263	.04022	1528	.0518	1609	.0582
6	.4001	.2263	.04022	1528	.0543	1609	.0618
7	.4002	.2264	.04026	1530	.0563	1610	.0641
8	.4003	.2258	.04004	1522	.0542	1602	.0614
9	.4003	.2264	.04026	1530	.0545	1610	.0627
10	.4003	.2265	.04029	1531	.0539	1612	.0618
11	.4005	.2255	.03994	1518	.0549	1598	.0628
12	.4000	.2262	.04018	1527	.0539	1607	.0619
13	.4004	.2270	.04047	1538	.0549	1619	.0623
14	.4000	.2258	.04004	1522	.0544	1602	.0615
15	.4002	.2260	.04012	1525	.0525	1605	.0615
Average					.0542		.0619

NOTES ON TABLE 21.—This table shows the results for copper cylinders, precompressed at 38 000 pounds per square inch, which had been held in an electric oven at 100° C for 28 days and then compressed at 40 000 pounds per square inch. The final total set in this case was 0.0619 inch.

TABLE 22.—Compression Tests Relating to Experiments on Aging of Copper Cylinders

L_1=length of cylinders after applying pressure of 38 000 lbs./in.² for about 2½ seconds.
L_2=length of cylinders after applying pressure of 40 000 lbs./in.² for about 2½ seconds, the pressure of 40 000 lbs./in.² being applied after aging cylinders at 0° and 100° C alternately during 28 days.

No.	L	D	A	Load equivalent to 38 000 lbs./in.²	$L-L_1$	Load equivalent to 40 000 lbs./in.²	$L-L_2$
	Inch	Inch	Inch²	Pounds	Inch	Pounds	Inch
1	0.4004	0.2260	0.04012	1525	0.0558	1605	0.0636
2	.4001	.2259	.04008	1523	.0556	1603	.0630
3	.4001	.2260	.04012	1525	.0544	1605	.0624
4	.4004	.2263	.04022	1528	.0548	1609	.0613
5	.4001	.2261	.04015	1526	.0543	1606	.0624
6	.4002	.2260	.04012	1525	.0552	1605	.0637
7	.4000	.2259	.04008	1523	.0525	1603	.0593
8	.4001	.2268	.04040	1535	.0539	1616	.0626
9	.4003	.2258	.04004	1522	.0540	1602	.0624
10	.4002	.2267	.04036	1534	.0541	1614	.0618
11	.4002	.2259	.04008	1523	.0530	1603	.0614
12	.4000	.2267	.04036	1534	.0538	1614	.0609
13	.4005	.2254	.03990	1516	.0546	1596	.0622
14	.4000	.2257	.04001	1520	.0538	1600	.0615
15	.4005	.2261	.04015	1526	.0505	1606	.0575
Average					.0540		.0617

NOTES ON TABLE 22.—This table indicates that the final total set of 0.0617 inch was obtained on cylinders precompressed at 38 000 pounds per square inch and aged at 0 and 100° C, alternately, during 28 days and then compressed at 40 000 pounds per square inch.

TABLE 23.—Compression Tests Relating to the Experiments on Aging of Copper Cylinders

L_1=length of cylinders after applying pressure of 38 000 lbs./in.² for about 2½ seconds.
L_2=length of cylinders after applying pressure of 40 000 lbs./in.² for about 2½ seconds, the pressure of 40 000 lbs./in.² being applied after aging cylinders at room temperature for 30 days.

No.	L	D	A	Load equivalent to 38 000 lbs./in.²	$L-L_1$	Load equivalent to 40 000 lbs./in.²	$L-L_2$
	Inch	Inch	Inch²	Pounds	Inch	Pounds	Inch
1	0.4008	0.2258	0.04004	1522	0.0506	1602	0.0537
2	.4008	.2257	.04001	1520	.0561	1600	.0636
3	.4005	.2257	.04001	1520	.0558	1600	.0641
4	.4007	.2260	.04012	1525	.0560	1605	.0617
5	.4007	.2262	.04018	1527	.0550	1607	.0612
6	.4008	.2259	.04008	1523	.0550	1603	.0628
7	.4008	.2260	.04012	1525	.0537	1605	.0613
8	.4005	.2259	.04008	1523	.0527	1603	.0602
9	.4010	.2260	.04012	1525	.0552	1605	.0616
10	.4008	.2259	.04008	1523	.0555	1603	.0624
11	.4011	.2262	.04018	1527	.0553	1607	.0621
12	.4006	.2267	.04036	1534	.0547	1614	.0618
Average					.0546		.0614

FIG. 7.—*Longitudinal section of copper cylinder annealed at 650° C.* × *100*

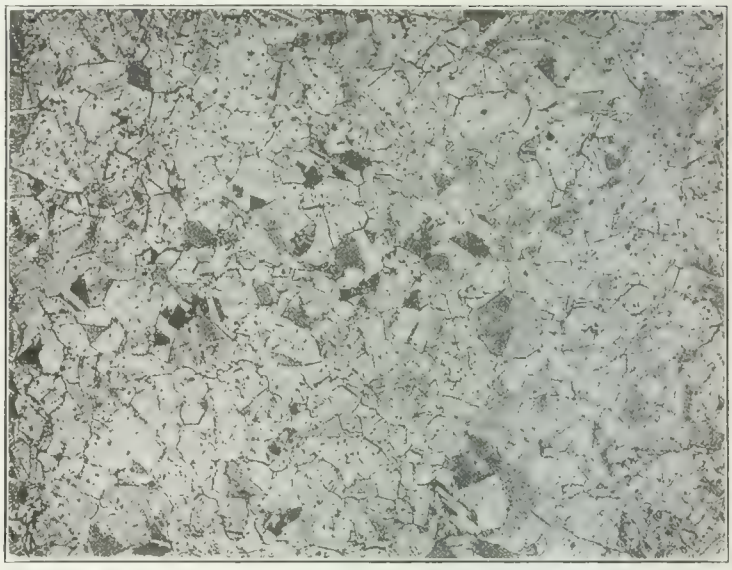

FIG. 8.—*Cross section of copper cylinder annealed at 650° C.* × 100

FIG. 9.—*Longitudinal section of copper cylinder annealed at 650° C.* × 100

TABLE 24.—Compression Tests Relating to the Experiments on Aging of Copper Cylinders

L_1=length of cylinders after applying pressure of 38 000 lbs./in.^{2} for about 2½ seconds.
L_2=length of cylinders after applying pressure of 40 000 lbs./in.^{2} for about 2½ seconds, the pressure of 40 000 lbs./in.^{2} being applied after aging cylinders at room temperature for 27 days.

No.	L	C	A	Load equivalent to 38 000 lbs./in.^{2}	$L-L_1$	Load equivalent to 40 000 lbs./in.^{2}	$L-L_2$
	Inch	Inch	Inch^2	Pounds	Inch	Pounds	Inch
1	0.4000	0.2268	0.04040	1535	0.0547	1616	0.0630
2	.4001	.2260	.04012	1525	.....	.....	.....
3	.3999	.2260	.04012	1525	.0525	1605	.0592
4	.3996	.2258	.04004	1522	.0550	1602	.0621
5	.3996	.2265	.04029	1531	.0529	1612	.0609
6	.4001	.2259	.04008	1523	.0524	1603	.0588
7	.4001	.2257	.04001	1520	.0551	1600	.0626
8	.4001	.2256	.03997	1519	.0544	1599	.0618
9	.3996	.2258	.04004	1522	.0561	1602	.0618
10	.4002	.2260	.04012	1525	.0547	1605	.0621
11	.3999	.2255	.03994	1518	.0541	1598	.0621
12	.3998	.2261	.04015	1526	.0548	1606	.0628
Average	.....	.....	.....	.....	.0542	.....	.0616

NOTE ON TABLES 23 AND 24.—Cylinders precompressed at 38 000 pounds per square inch were aged at room temperature for 30 days and then compressed at 40 000 pounds per square inch. The mean total sets were 0.0614 and 0.0616 inch, respectively.

Intercomparing all results stated above (Table 25) it may be concluded that aging at temperatures of 0–100° C makes the compressed copper softer. It should be noted, however, that in the author's opinion the data in regard to aging is not sufficient; such an important problem needs very careful additional experiments in static as well as impact and firing tests after different periods of aging. The above results also are contradictory to those of H. W. R. Mason,[12] who found from his impact experiments that in copper crushers spontaneous annealing does not take place.

[12] H. W. R. Mason, Resistance of Copper Crushers during Compression, Arms and Explosives; July 1, 1918.

TABLE 25.—Table Showing Average Results on Effect of Aging on Reduction in Length of Compressed Copper Cylinders

Reference to cylinders	Precompression at 38 000 lbs./in.²		Compression at 40 000 lbs./in.²	
	Date of precompression	Average total set of cylinders per inch	Date of compression	Average total set of cylinders per inch
		Inch		Inch
Table 14	June 26, 1919	0.0540	One hour later of same day, June 26, 1919.	0.0606
Table 20	Aug. 29, 1919	.0538	One hour later of same day, Aug. 29, 1919.	.0608
Table 21	July 30, 1919	.0542	Aug. 27, 1919, after aging at 100° C from July 30 to Aug. 27.	.0619
Table 22	do	.0540	Aug. 27, 1919, after aging at 0° C and 100° C alternately from July 30 to Aug. 27.	.0617
Table 23	do	.0546	Aug. 29, 1919, after aging at room temperature from July 30 to Aug. 29.	.0614
Table 24	July 31, 1919	.0542	Aug. 27, 1919, after aging at room temperature from July 31 to Aug. 27.	.0616

III. MICROSCOPIC EXAMINATION

Longitudinal and cross sections of copper cylinders were examined microscopically (Figs. 7–15). In all cases the cylinders were etched with NH_4OH and H_2O_2.

In order to show distortion of grains under pressure, some micrographs were taken of the same spots before and after compression—that is, the cylinder was etched, the micrograph was taken, and the spot marked. Then the cylinder was compressed at 30 000 pounds per square inch, and a micrograph of the marked spot was taken again. This is shown in Figs. 12–15, where (Figs. 13 and 15) many slip bands may be seen.

IV. SUMMARY

Several conclusions may be drawn from these experiments, which only confirm the already known phenomena, as well as those expected from the known properties of copper:

1. The length of copper crusher cylinders decreases considerably under repeated compressions of the same load. The relation between this decrease in length and number of times load was applied is nearly proportional within certain limits.

FIG. 10. *Longitudinal section of copper cylinder annealed at 540° C.* × 100

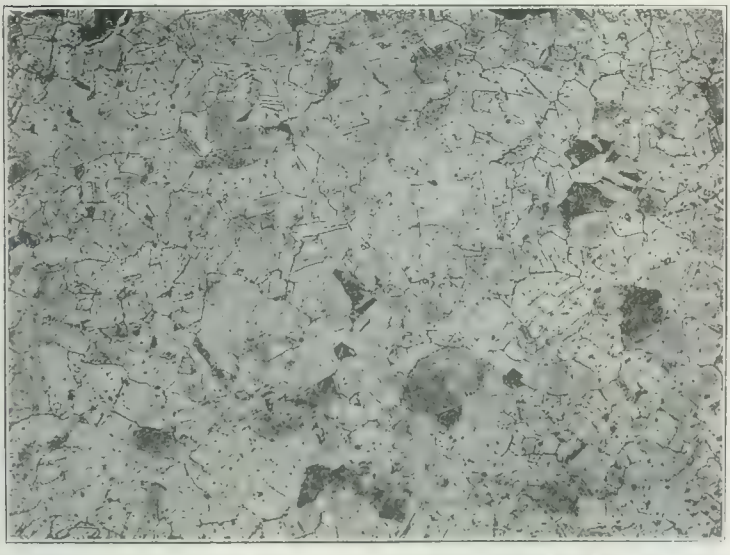

FIG. 11.—*Cross section of copper cylinder annealed at 540° C.* × 100

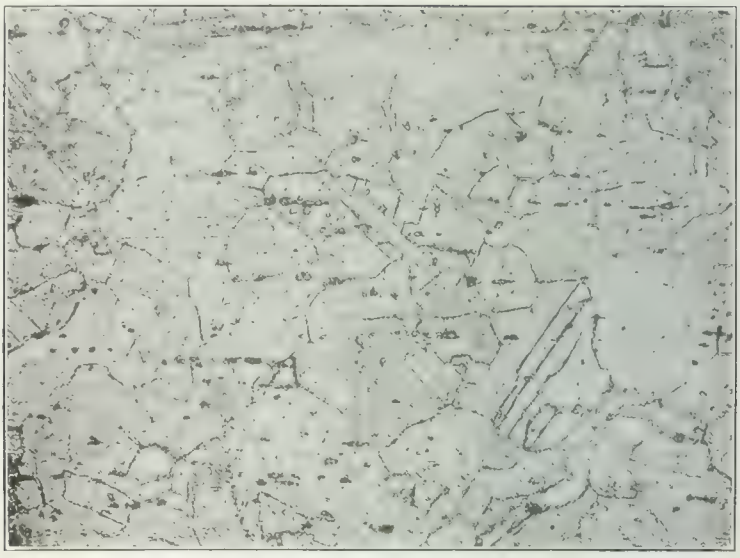

FIG. 12.—*Longitudinal section of copper cylinder annealed at 650° C. Micrograph represents the spot located midway bases and near to the axis of cylinder* × 200

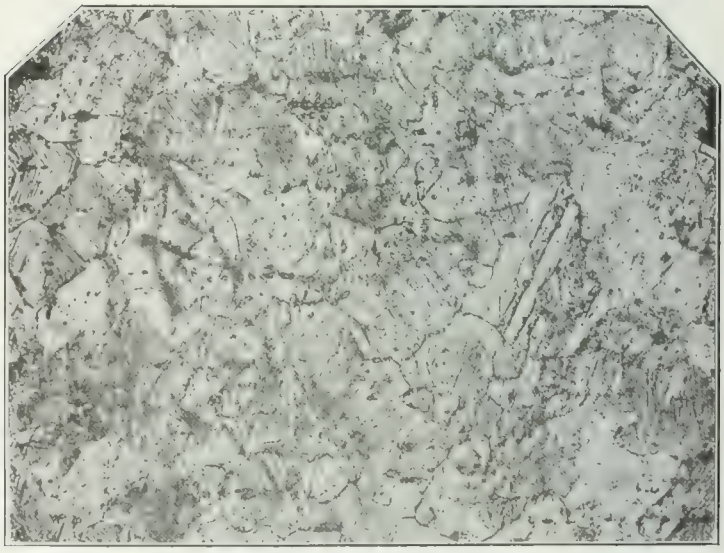

FIG. 13.—*Same cylinder and same spot as Fig. 12, but after compression at 30 000 lbs. per square inch.* × 200

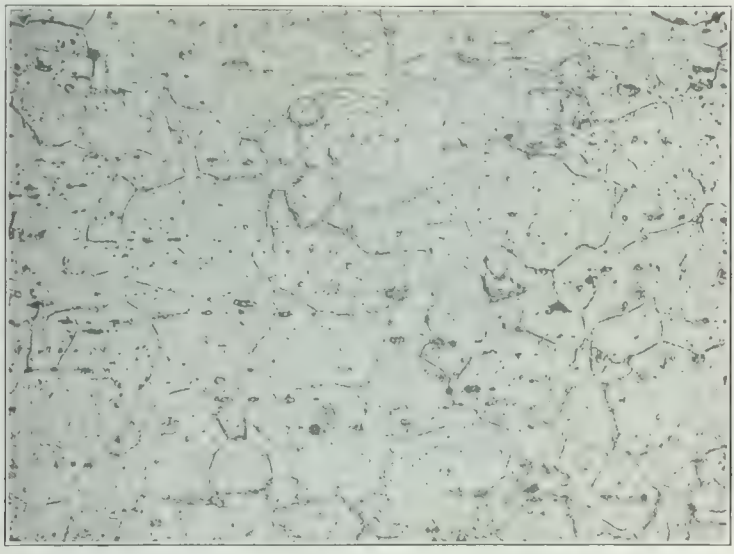

FIG. 14.—*Longitudinal section of copper cylinder annealed at 650° C. Spot is located near one of the bases of the cylinder and very near its periphery.* ×200

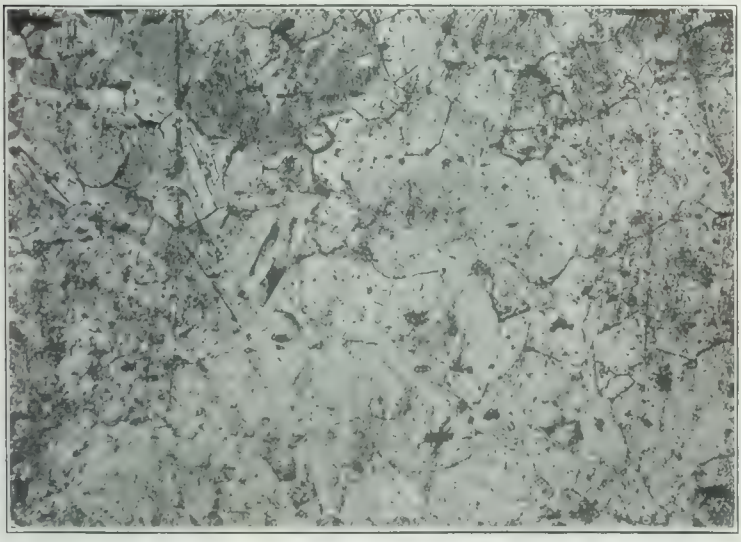

FIG. 15.—*Same cylinder and same spot as in Fig. 14, but after compression at 50000 lbs. per square inch* ×200

2. The change in length with repeated constant loads is greater with the greater load.

3. Repeated application of successive decreasing loads causes a gradual decrease in length within certain limits.

4. The length changes but slightly with longer application of load when the load is applied in such a manner as to occasion the maximum stress only for an instant.

5. Application of the same load for the same period (greater than about 3 seconds), but holding the beam balanced by means of one additional application (impulse) of the same load after the beam had started to drop, causes about twice the decrease in length observed in the previous case, when the beam was allowed to drop after balance is obtained.

6. Double application of the same load for 2½ seconds each time, the second load being applied at some interval after the first load has been released, causes a decrease in length much greater than in the previous two cases. This holds true even when the pressure is applied for as long as 25 seconds in the case of paragraphs 4 and 5.

7. In case the last load is considerably greater than any previous loads, the change in length caused by the last load is practically independent of the previous loads—that is, it is the same as would be obtained by compressing a previously uncompressed cylinder.

8. When two successive loads of considerable amount (approximately 40 000 pounds per square inch) differing from one another by about 2 000 pounds per square inch, are applied, the second being greater than the first, the change in length due to the last load is considerably greater than that obtained where the pressure is applied on previously uncompressed cylinders, and this difference increases as the difference between the two loads successively applied decreases.

9. When several successive loads of considerable amount are applied, differing by about 2000 pounds per square inch, each greater than the preceding load, the total change in length of the cylinders due to the last compression increases with the number of loads previously applied.

10. Copper cylinders annealed at 1200° F 650° C are softer than those annealed at 1000° F 540° C.

11. It is probable that aging at temperatures within 0–100° C softens the compressed copper somewhat.

12. Concerning the use of precompressed or uncompressed copper cylinders the conclusion from these experiments may be drawn for cases when the checking of every copper cylinder by precompression is necessary, a precompression of the cylinders at a pressure of at least 8000 pounds per square inch below the expected maximum pressure can be employed without impairing the ability of the precompressed cylinders to register the maximum pressure in equally reliable manner as an uncompressed cylinder would.

WASHINGTON, October 15, 1920.

DEPARTMENT OF COMMERCE

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No. 186

OSCILLOGRAPH MEASUREMENTS OF THE INSTANTANEOUS VALUES OF CURRENT AND VOLTAGE IN THE BATTERY CIRCUIT OF AUTOMOBILES

BY

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Bureau of Standards

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OSCILLOGRAPH MEASUREMENTS OF THE INSTANTANEOUS VALUES OF CURRENT AND VOLTAGE IN THE BATTERY CIRCUIT OF AUTOMOBILES

By G. W. Vinal and C. L. Snyder

ABSTRACT

This investigation was begun to study the demands upon starting and lighting batteries in the operation of various types of automobiles. By using an oscillograph, photographic records of the current and voltage were obtained. In addition to the data relative to the battery requirements, the interpretation of the records has indicated the possibility of using this method for the study of lubrication, and engine problems relating to torque, speed, friction, ignition, compression, and distributor action. Oscillograms obtained for five of the cars measured are shown in the illustrations. The results given are intended to be suggestive rather than quantitative measurements of performance.

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1. INTRODUCTION

For some months, at the request of the Motor Transport Corps of the War Department, the Bureau of Standards has been working on specifications for starting and lighting batteries. These specifications have now been completed and will be published as the appendix to a circular to be issued by this Bureau. Various laboratory tests have been made also on batteries of this class. In addition to these tests it seemed desirable to study the performance of batteries when in actual use on automobiles. Measurements of current and voltage of the batteries as made by the ordinary indicating instruments do not give accurate or adequate information because of the rapidly fluctuating values of these quantities. We have, therefore, measured the instantaneous values of current and voltage from oscillograph records. The

variations of voltage and current when the engine is operated by the starter, and the variation of charging current through the battery under running conditions, are shown in the oscillograms that follow.

In the course of the experiments additional information was obtained with respect to the operation of the starter system and the engine itself. It is intended, therefore, in this paper to point out some of the effects relating to torque, temperature, compression, lubrication, distributor action, and flywheel velocity, in addition to the battery characteristics. The results which are given are suggestive only of the possible application of this method for the study of the performance of starter systems and internal-combustion engines. It is believed that a development of this method may provide an easy and exact method for studying the operation of various parts. This method was found entirely satisfactory for studying the problems connected with the battery.

Experiments have been made on a number of different cars. From the records obtained, we have chosen those of five cars which are designated in this paper as A, B, C, D, and E. Each of these cars was of a well-known make. For purposes of comparison, we have given in Table 1 data as to the history and characteristics of these cars which is essential to a proper understanding of the curves. All of the cars were in ordinary running condition at the time of the test.

TABLE 1.—Designation and Data of the Cars Tested

Designation of car	Year	Cylinders	Miles driven	Bore	Stroke	Starter system	Battery voltage
A	1916	4	15 000	3⅞	4½	Single unit	12
B	1920	4	5 000	3¾	4	Two unit	6
C	1917	6	12 000	3¼	4½	do	6
D	1917	4	20 000	3⅞	5	do	6
E	1919	8	18 000	3⅛	5⅛	Single unit	6

2. METHODS AND APPARATUS

The instantaneous values of current and voltage were obtained by means of a three-element moving-coil type oscillograph. In order to obtain records for periods of long duration, the ordinary film drum was replaced by a camera of special construction in which bromide paper in rolls 100 feet long was used. When

making a record, the sensitized paper was wound upon a drum in the lower end of the camera after having passed the oscillograph slot through which the recording images were projected. One of the elements was used to record the voltage at the terminals of the battery, another the current through the battery circuit, and the third the time record, which consisted of the half-second ticks of a chronometer.

The camera was arranged so that the photographic paper moved at a uniform speed, irrespective of the amount of paper remaining in the camera. It was possible, however, to change the speed of the paper at will. Some of the records have been made with the paper moving at low speeds, about 2 cm per second, others at an intermediate speed of 3½ cm per second, but most of them at high speed, 7½ cm per second.

In order to measure the current through the battery circuit, a shunt of 0.006 of an ohm resistance was inserted in the circuit at the battery. Potential wires from this shunt were connected with the current element in the oscillograph. Measurements of the current and voltage were also made simultaneously by an ammeter and voltmeter. These meters have too slow a period to give the initial values. The meter readings, however, gave satisfactory average values for the current and voltage when the starter was operated at a uniform rate. Such readings were used as a check upon the calibration of the elements in the oscillograph. The deflections of the elements were approximately proportional to the current or voltage measured. In some cases, however, a deviation was found, which may have been due to the slight tension of the element. The tension was necessarily reduced to a minimum in order to give sufficient deflection. It was found necessary to calibrate the current and voltage elements with steady currents and voltages which could be accurately measured. This calibration was necessarily repeated each time the tension of any element was altered, or when the gold fuses were replaced. It will be noted that the half-second ticks of the time record are unequal. This was due to the action of the relay in the circuit. From the beginning of one second to the beginning of the next second, however, can be considered as one second to a very high degree of accuracy.

3. VARIATIONS IN VOLTAGE AND CURRENT

The general character of the curves [1] obtained is shown in the figures following. All of these oscillograms begin at the right and are, therefore, to be read from right to left. A more detailed examination of these and similar figures will be given in the later

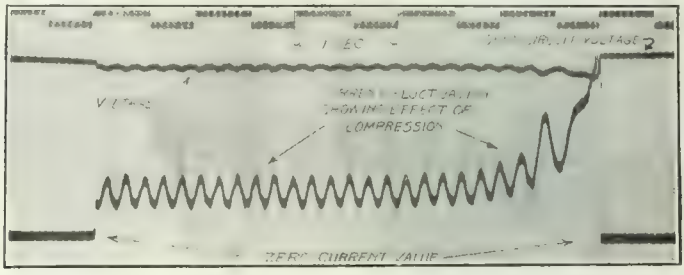

FIG. 1.—*Record of current and voltage in the battery circuit of car A when the starter was cranking the engine without ignition*

The fluctuations in these curves show the effect of compression in the cylinders. The initial current was approximately 125 amperes. The time scale in half-seconds is shown at the top of the record

sections. Fig. 1 shows curves for car A. The time intervals are recorded at the top of the record. The two curves below represent the fluctuations of the voltage and current of the battery when the starter is in operation. The successive peaks of the latter show the increase of current for each compressive stroke. Fig. 2 shows the operation of the same system with the ignition. It will be

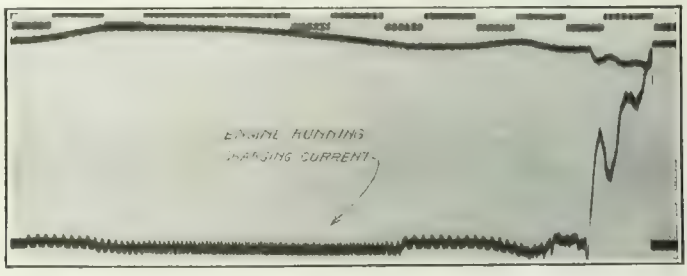

FIG. 2.—*Record of the same car as Fig. 1 when the engine was running*

The engine started on the second compression. The voltage curve falls at the beginning, but rises to higher values as soon as the charging current begins. The charging current lies slightly below the zero current value at the ends of the record

seen from the curve that the engine started on the second compression somewhat less than one second after the starter switch was closed. The variations in the current curve will be explained under the head of section 6. Figs. 3 and 4 show similar observa-

[1] Vincent, Trans. Soc. Auto. Eng., 11, part 1, p. 194, 1916 has derived mechanically the general shape of the torque curves for 6, 8, and 12 cylinder engines. The current curves of our oscillograms are proportional to the torque and show actual performance.

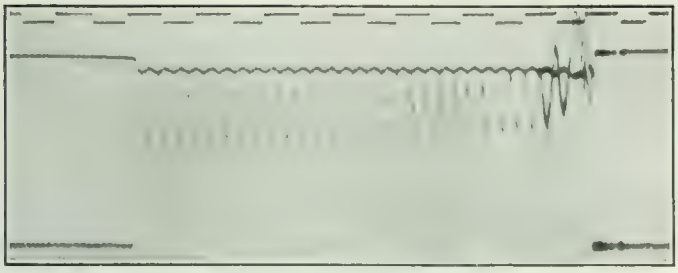

FIG. 3.—*Record of current and voltage in the battery circuit of car B operated without the ignition*

The current fluctuated rapidly during the first half-second through a range of more than 100 amperes, the maximum being about 300 amperes. The minimum values of voltage correspond to the maximum values of current

FIG. 4.—*Record of car B with engine starting*

The differences in operation between a two-unit and a single-unit system may be seen by comparing this figure with Fig. 2. The point at which the starter pedal was released can be seen

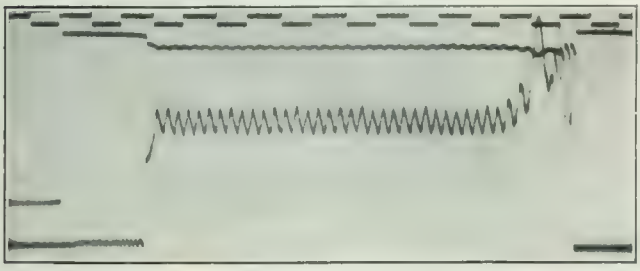

FIG. 5.—*Record of current and voltage in battery circuit of car C operated without the ignition*

The current fluctuated rapidly during the first half-second

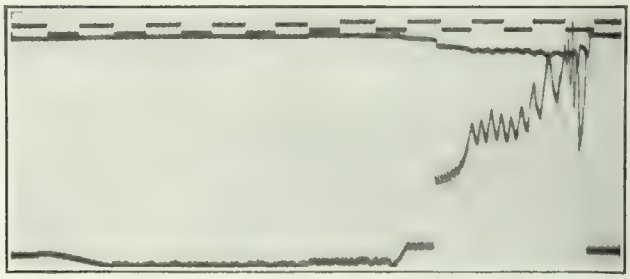

FIG. 6.—*Record of car C with ignition*

The engine started in about 2 seconds

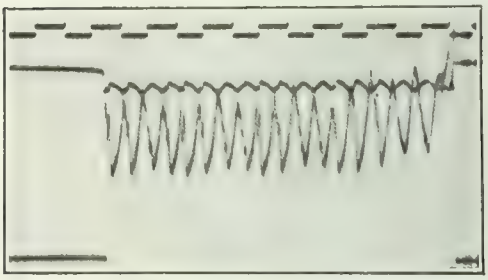

FIG. 7.—*Record of current and voltage in the battery circuit of car D operated without the ignition*

The initial current was about 250 amperes, the average running current about 125 amperes

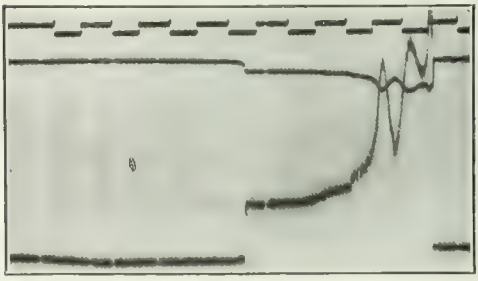

FIG. 8.—*Record for the same car as Fig. 7 with the ignition*

The engine started in about 1 second

tions made on car B. The initial current values of both Figs. 3 and 4 are beyond the limits of the paper. During the first half second the current fluctuated violently between 200 and 300 amperes. In Fig. 3 it may be seen that the minimum voltage values correspond with the maximum current values. This is

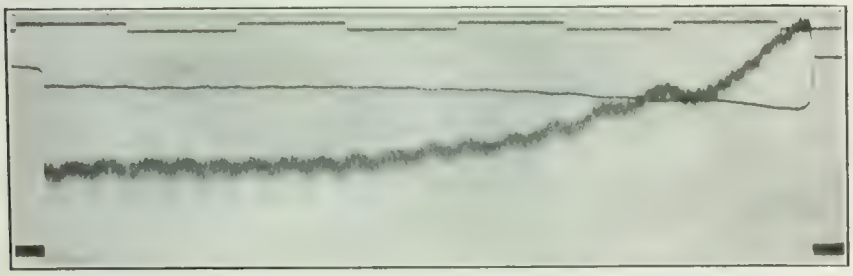

FIG. 9.—*Record of current and voltage in the battery circuit of car E as the starter cranks the engine, without the ignition*

The time scale of this and succeeding figures has been enlarged. The initial current was about 425 amperes. By comparing this figure with Fig. 10, in which only one bank of this engine was used, the overlapping effect of the cylinders may be seen

true of the other illustrations also, but in most of them it is not as clearly seen as here. The motor was started with the ignition on the battery in Fig. 4. The discontinuous point in the current curve shows the time at which the starter pedal was released. A corresponding rise in the voltage is to be observed. Figs. 5 and 6 were obtained on car C. The characteristics shown are somewhat

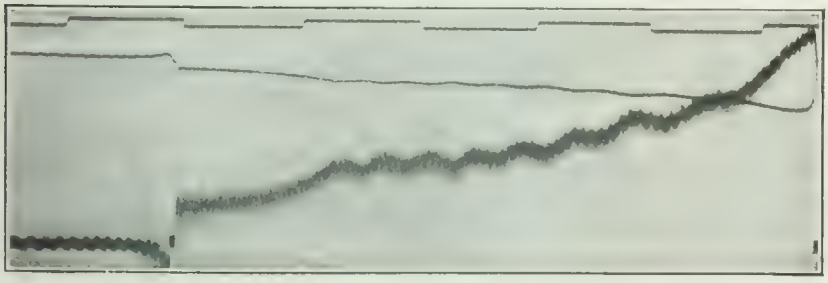

FIG. 10.—*Record similar to Fig. 9 but with the ignition*

A point of zero value in the current curve after the engine started indicates the operation of the reverse-current relay

similar to those in Figs. 3 and 4. Figs. 7 and 8 were obtained on car D. These curves are without the violent fluctuations observed during the first half second in the case of the four preceding curves. Figs. 9 and 10 show curves for car E. The initial current was approximately 425 amperes, decreasing to an average running value of about 150 amperes, as shown in Fig. 9. In Fig. 10 the

ignition switch was closed and the engine started at the end of two and one-fourth seconds.

In the illustrations for the various cars, characteristic differences are apparent in the current and voltage curves. A comparison of these curves, shown in Figs. 1, 3, 5, 7, and 9, indicates that some are more nearly sine curves than others. These differences are probably characteristic of differences in the designs of the engines. By making a harmonic analysis of such curves, it might be possible to trace the effects of the timing of the valves and other factors. For such an analysis it would be desirable to increase the time scale as has been done in Figs. 11, 12, and 13. We have not attempted such a harmonic analysis in this paper.

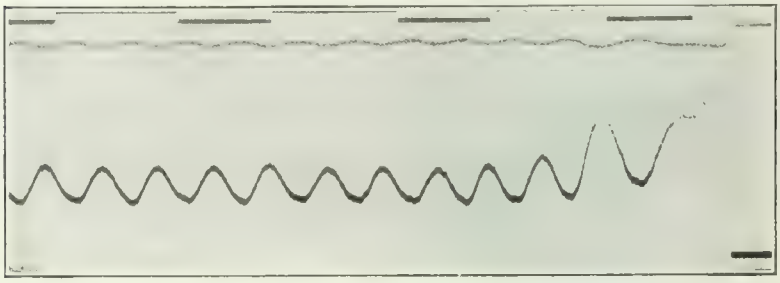

FIG. 11. *Record from car A at radiator temperature 20° C*

The readings taken from these curves are given in Table 2

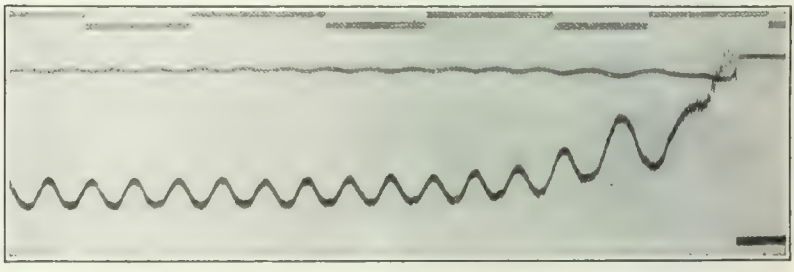

FIG. 12.—*Record as for Fig. 11 but at radiator temperature of 50° C*

The readings taken from these curves are given in Table 2

Figs. 2, 4, 6, 8, and 10 show characteristic differences in the starter systems, particularly between the single and two-unit systems.

The sensibility of the oscillograph was not the same for all of the measurements, and some differences in appearance of the curves have been caused by variations in the intensity and width of the beam of light. The cars were all in ordinary running condition.

4. EFFECT OF TEMPERATURE

Figs. 11, 12, and 13 show the effect of the temperature of the engine on the current and voltage required for starting. The engine was operated by the starter without ignition. These curves were obtained on car A at radiator temperatures of 20, 50, and 77° C (68, 122, and 171° F), respectively. The initial values for current and voltage have been read from these curves and tabulated in Table 2, together with the values obtained when the speed was constant. The average operating values of current and voltage at constant speed as read from the curves agree closely with the values given by the indicating instruments in the circuit. The difference between the maximum and minimum values at constant speed is the amplitude of column 9 in the table. It will

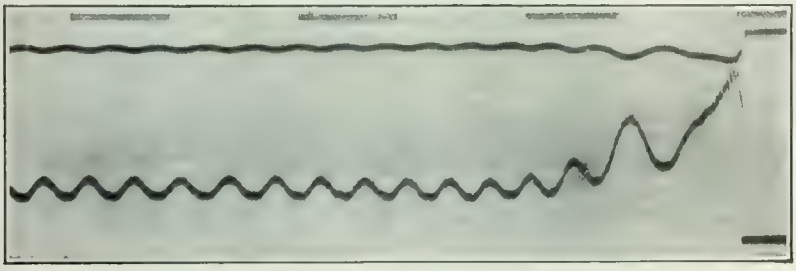

FIG. 13.—*Record as for Figs. 11 and 12, but at radiator temperature of 77° C*

The readings taken from these curves are given in Table 2

be seen by comparing the results for this car at 20° C with those at higher temperatures that the initial starting current does not decrease in the same proportion as the operating current, when the temperature is increased. This indicates that the initial current which the battery must supply depends more on the inertia of the system than on the temperature of the engine. The average operating current at 50° C is practically the same as the current at 77° C. A change is to be noted, however, in the amplitude of the fluctuations. At 20° C the amplitude is 36 amperes, at 50° C it is 28 amperes, and at 77° C it is 16 amperes. The significance of these changes will be brought out more clearly later on. On examination of these curves with reference to the time scale, it will be seen that the speed of the engine may be easily computed since there are two compressions for each revolution of the crank shaft. The values for the car E given in Table 2 have been computed from 11 oscillograms, which are not shown as illustrations. The power required initially to start the system in motion and the

average operating power when the speed was constant show little change due to temperature above 35° C. It will be noticed in the table that slightly more power is required initially to start the system when the throttle is closed, but that the average operating power is less if the speed remains constant. The latter result may be due to effects produced by the greater compression in the cylinders when the throttle is open. As these observations were made in the summer it was not convenient to obtain temperatures below those given in the table. The power required to overcome the inertia of the system is somewhat less than the lowest figure given in column 10 of Table 2. The power required to overcome the compression is practically constant, irrespective of the temperature, provided the throttle opening is fixed. The power to overcome friction would probably increase rapidly at temperatures below those recorded here, as is indicated by Fig. 20. The increase in average operating power required for car E at the highest temperatures suggests increased friction. The engine had been run for a considerable period of time to increase the temperature. The oil and bearings had become heated and it is probable that the bearings were tighter at the higher temperatures because the engine had just been overhauled. Similar effects might also be produced by insufficient lubrication at these temperatures.

TABLE 2.—Instantaneous and Average Values of Current and Voltage Read from Figs. 11, 12, and 13 for the Car A and from Other Oscillograms Which Are Not Shown for the Car E

Car	Temperature of radiator water [a]	Engine speed	Throttle	Initial values		Average operating values		Amplitude of current	Power required	
				Current	Voltage	Current	Voltage		Initial	Average operating
	° C	r p m		Amp.	Volts	Amp.	Volts	Amp.	Watts	Watts
A	20	116.8	Closed	136	10.3	62	11.1	36	1400	688
A	50	162.6	do	127	11.1	38	11.8	28	1410	449
A	77	150.9	do	117	11.1	40	11.8	18	1300	472
E	25	89.0	Closed [b]	455	4.64	133	5.61	.....	2110	746
E	25	90.0	Open [b]	449	4.62	137	5.62	.....	2073	770
E	35	91.6	Closed	400	4.57	136	5.38	.....	1826	732
E	35	94.6	Open [b]	409	4.44	140	5.36	.....	1814	752
E	65	93.7	do	417	4.46	138	5.37	.....	1860	742
E	69	100.0	Closed	432	4.56	148	5.40	.....	1984	800
E	82	96.0	Open	417	4.42	154	5.38	.....	1842	829

[a] Atmospheric temperature for measurements on the car A, 25° C; for the car E, 28° C.

[b] Observations are the mean of two determinations.

5. EFFECT OF COMPRESSION AND LUBRICATION

In order to clearly demonstrate that the fluctuations in the current curve are due to compression in the cylinders, a run was made in which one priming cup was removed. The results of this experiment on the car A are shown in Fig. 14. This illustration shows that there was only a slight compression in the cylinder that was open.

The power furnished by the battery is expended in several ways: First, there is a small part which is used up within the starter itself. This may be measured as in the case of any electric motor. A second part is expended in compressing the gas within the cylinders. A third part is expended in overcoming friction in the engine and

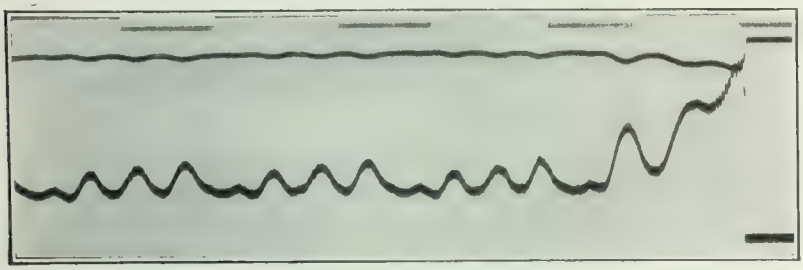

FIG. 14.—*Experiment to show the effect of compression on the record of current and voltage of the battery*

A pet cock was removed from one cylinder, car A. The compressive stroke in this cylinder was quicker than for the others, but the succeeding downward stroke occupied more time. As there was little compression in this cylinder no energy was returned to the system by compressed gas

varies with the speed and the lubrication. A fourth part is used for ignition on some cars. The amount of power which the battery furnishes may be obtained for any instant by multiplying the instantaneous values of current and voltage. If this product is also multiplied by an increment of time, we may obtain the total energy or work done by the battery during this instant of time.

It is apparent, therefore, that careful measurements of the electric power and time can furnish information as to the amount of work being done by the battery during any part of a cycle. One significant result of this may be seen in Fig. 14, where it will be observed that the engine speeds up in passing what would have been the compression stroke of the cylinder had the priming cup not been removed. As there was no compression in this cylinder, no energy could be returned to the system by compressed gas. The increased time required to reach the next maximum (Fig. 14) shows that the engine decreased in speed. This fact is determined

by measuring the distance between the successive maximum values of the curve. A rolling effect results, which may be noticed by the difference in height of the successive maximum values. The watts delivered by the battery may be used as a sensitive and exact method of determining the instantaneous speed of the flywheel. The greater the inertia of the flywheel the smaller will be the fluctuations of the current and voltage curves.

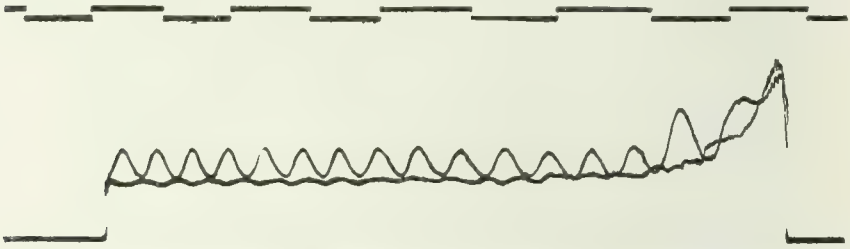

FIG. 15.—*Showing the effect of removing all spark plugs on the current curve, car A*

Two records have been superposed. Both were made with a radiator temperature of 20° C. For one the spark plugs were in place; for the other they were all removed

The curves of Figs. 11, 12, and 13 (car A), show a decreased amplitude at the higher temperatures. This suggests the conclusion that the amplitude for this type of engine depends on lubrication as well as compression. The effect due to friction is least at the top and bottom of the stroke and greatest when the crank arm and the connecting rod are at right angles. This result has been confirmed by experiments using one bank of the engine of car E.

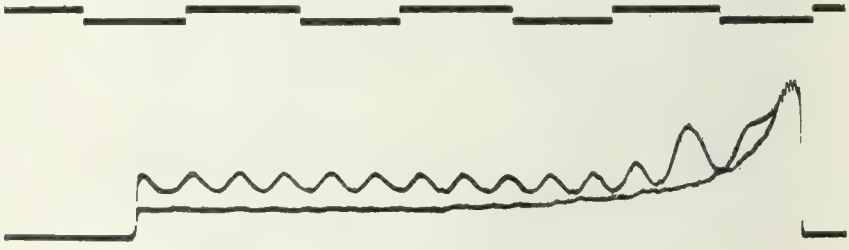

FIG. 16.—*Same as Fig. 15 but at radiator temperature of 47° C*

The curve for plugs out is lower than in Fig. 15 and higher than in Fig. 17

It is possible to separate the effect of compression from that of friction and other losses by relieving the compression in all of the cylinders. This might best be done by removing the head of the engine, but in these experiments the compression was more conveniently relieved by removing the spark plugs. A series of comparative measurements, the results of which are shown in Figs.

15, 16, and 17, were made on the car A at three different radiator temperatures, 20, 47, and 84° C respectively. These figures show that the curves obtained when the spark plugs were removed are devoid of the marked fluctuations of current observed when the spark plugs were in place. At each temperature a record of the current values was made with the spark plugs in, followed by an

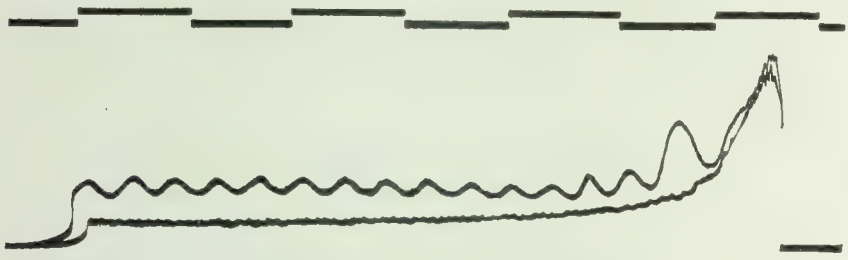

FIG. 17.—*Same as Figs. 15 and 16 but at radiator temperature of 84° C*

Values read from Figs. 15, 16, and 17 are used in Fig

exactly similar measurement with the spark plugs removed. As there was no combustion in the cylinders the temperature of the radiator water is a fair indication of the temperature of the walls of the cylinders. The corresponding curves for each temperature have been superposed and traced together in Figs. 15, 16, and 17, care being taken to make them in perfect register, both as to the beginning of the curves and their position with respect to

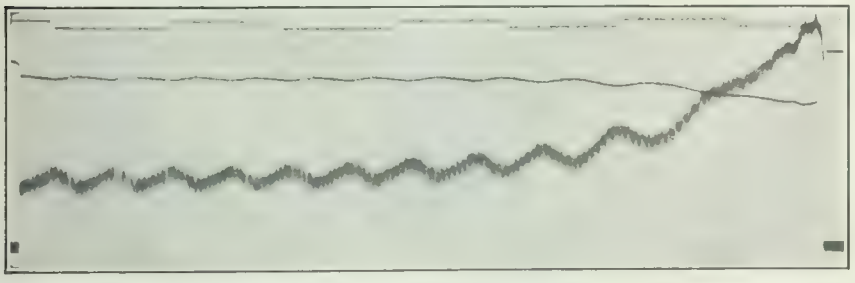

FIG. 18.—*Curves of current and voltage, car E, having all spark plugs removed from one bank of the engine*

Comparison of this with Fig. 9 shows the overlapping power strokes when all cylinders are in u

the horizontal axis. The results obtained at 20° C show that the curve without compression is practically tangent to the lower values of current when the spark plugs are in position. At the higher temperatures this curve falls somewhat below the minimum current values. Fig. 18 shows the results of an experiment on the car E with the spark plugs removed from the left bank. By

comparing this with Fig. 9 the overlapping effect of the cylinders when all are in use may be seen. Fig. 19 shows the results of an experiment on the car E from which all spark plugs had been removed. The same high value of the initial current is to be noted in this figure as in Figs. 9 and 18. If Figs. 18 and 19 were superposed, the current curve of Fig. 19 would be tangent to the bottom of the current curve of Fig. 18.

The solid curves given in Fig. 20 show the relation between the temperature and viscosity of the oil used in car A. The values are plotted in terms of absolute units of viscosity and also in more familiar units, Saybolt seconds. The dotted curve gives the relation between the temperature and current in turning the engine after the inertia of the system had been overcome. These data were obtained from Figs. 15, 16, 17. The current is proportional

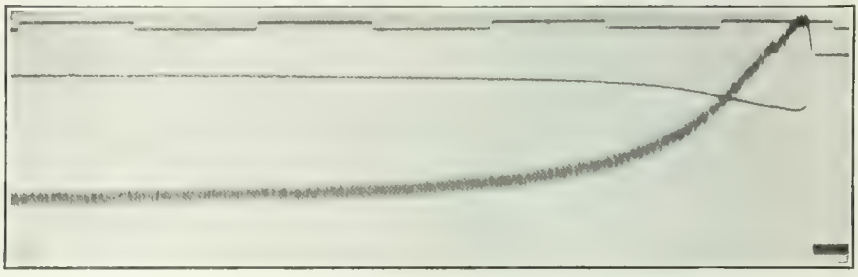

FIG. 19. *Curve of current and voltage, car E, having all spark plugs removed*

The initial value of current in this figure is about 425 amperes, which is the same as for Fig. 9 when all the spark plugs were in place

to the torque. These curves are plotted to an arbitrary scale. The marked similarity suggests a decrease in friction with increasing temperature. These curves show a proportionality to exist between the viscosity of the oil and the power required of the battery to turn the engine.

A number of experiments were made on the car A at temperatures of 20, 50, and 80° C, to determine the power consumed by the starter system for various throttle openings. In Fig. 21 we have plotted the values read from the oscillograms for power and engine speed plotted against an arbitrary scale of throttle openings. We have also plotted the compression curve as measured by a gage for the same throttle openings as well as these could be determined without the use of fixed orifices in the carburetor.

It appears that if the amount of energy lost in the starter itself is known and deducted and the compression released by removing

either the spark plugs or the head of the engine, the electrical power necessary to overcome friction may be calculated. In this way comparative measurements of the lubricating properties of different oils and friction losses in various parts of the machine might be made under running conditions.

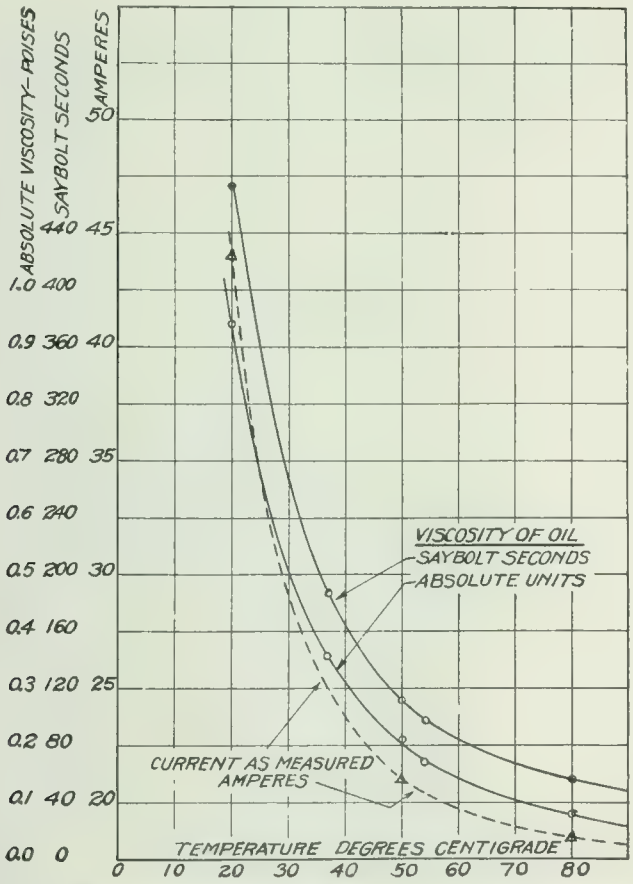

FIG. 20. *Relation of electric current in starter circuit to viscosity of the oil used in the engine*

The currents measured in amperes as given by the curves of Figs. 15, 16, and 17 spark plugs removed, have been plotted against the temperature. The viscosity of the oil in absolute units has been plotted for comparison. The similarity indicates a proportionality between the viscosity and the current. The viscosity expressed in Saybolt seconds is given also

At this point it may be interesting to examine a curve which was obtained when one of the four cylinders of the car A was leaking. The exact cause of the leak is not known, but is supposed to have been due to a particle of carbon lodged in an exhaust valve. A curve showing the result is given in Fig. 22, in which it

will be noted that every fourth maximum is considerably lower than the others.

Under this head there remains to be described one peculiar phenomenon which was observed in the experiment with the spark plugs out. It was observed in some cases where air was drawn directly into the cylinder, either by removing the spark plugs or taking out the priming cups, that greater demands were made on the battery in order to operate the starter after the spark

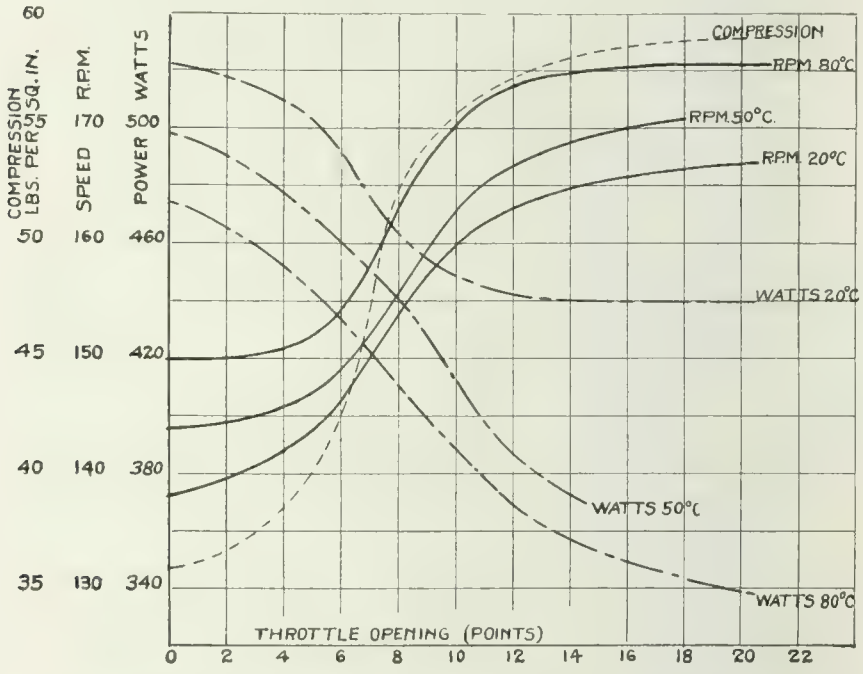

FIG. 21.—*Relation between throttle opening and electric power in watts delivered by the battery in cranking the engine of car A at 20, 50, and 80° C. The speed in r p m is also shown*

The dotted curve represents the compression in the cylinders measured by a gage for the same throttle openings as nearly as these could be repeated with the hand lever. The scale of throttle openings is arbitrary

plugs had been again replaced; that is to say, more work was required to turn the engine over. In Fig. 23 this effect is shown by the shape of the current curve. This record was made with a radiator temperature of 47° C, immediately after the spark plugs had been replaced. The curve shows the initial effort of the battery to have been exceeded by a second maximum which lasted for more than a half second, during which the engine did not turn. After gradually passing this maximum, the normal operation was restored. The same phenomenon is to be noticed in another curve, Fig. 24, which was made immediately after an experiment

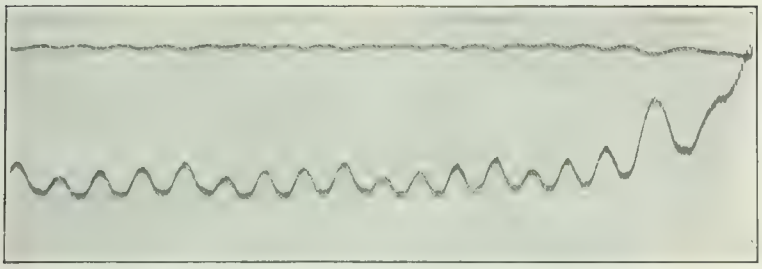

FIG. 22.—*Effect of a partly open valve in one cylinder, car A, ignition off*

FIG. 23. —*Record showing a sticking of the engine after drawing air directly into the cylinders.*

The spark plugs had been removed for the experiments of Fig. 16 and replaced. Temperature, 47° Cs. The starter was unable to turn the engine for about 1 second

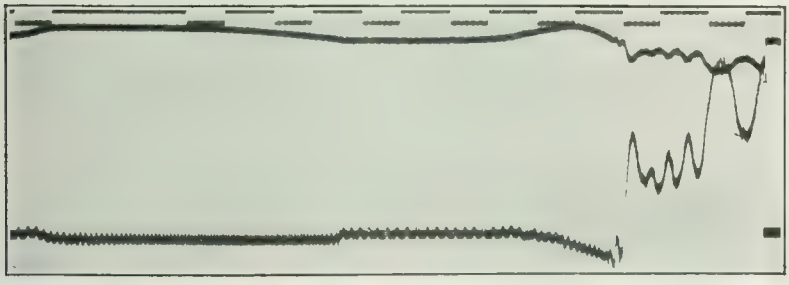

FIG. 24. *Effect similar to that of Fig. 23, following the experiment of Fig. 14*

The pet cock had been replaced. The sticking effect at the first compression was repeated at the fifth

in which a priming cup was removed from one cylinder. Here again it is observed that the first compression stroke, viz, that of the cylinder from which the priming cup had been removed and replaced, required as much power as was initially expended by the battery in starting the system in motion. The gas which was contained in the cylinder at this time was, of course, expelled by the succeeding stroke, but an excessive power consumption was again observed on the next compression in this cylinder as shown by the fourth maximum. The ignition switch was on during this experiment, and the engine began to operate on the next stroke, hence it is not possible to tell whether this effect persisted. The cause of this peculiar characteristic is not known.

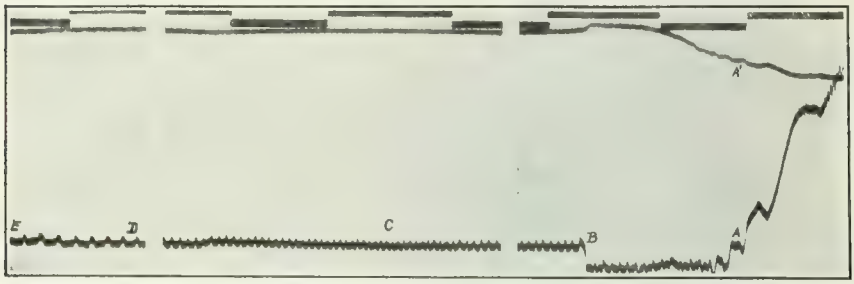

FIG. 25.—*The starting of the engine of car A and characteristics of the charging current*

A and *A'* represent the operation of the reverse-current relay; *B*, starter pedal released; *C*, engine speed increased; decreased at *D*; ignition cut off at *E*. Interruptions in charging current are due to contacts of ignition system

6. CHARACTERISTICS WITH THE ENGINE RUNNING

Experiments upon the car A similar to those shown in Figs. 11, 12, and 13 have been made with ignition on. One of the curves which has been thus obtained is shown in Fig. 25. In Fig. 25 the initial value of the current recorded is 136 amperes, which is exactly equal to that of Fig. 11. The initial value of the voltage, 10.3 volts, is also the same. The engine started on the second compression and the current values fell below the initial line, which shows that the battery was being charged. An immediate rise in the voltage curve to a point above open-circuit voltage shows this to be true. At the point *A* on the current curve and a corresponding point *A'* on the voltage curve is a short horizontal region where it is probable that the reverse-current relay operated. The charging voltage rises to 2.7 volts per cell, or 16.4 volts for the battery as a whole. This high charging voltage was due to the high rate at which the battery was

being charged during the time that the starter switch was still closed. At the point marked *B* the starter switch was released. An immediate decrease in the charging voltage and the charging current was observed. The charging current decreased at regular intervals, the spacing of which varied with the speed of the engine. The engine was racing at *C* and slowed down at *D*. The ignition

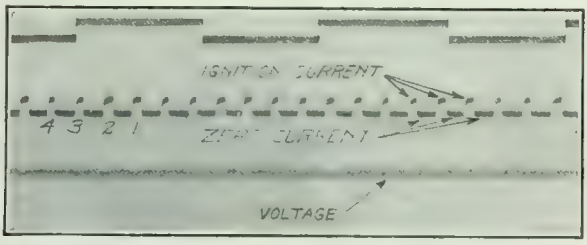

FIG. 26.—*Ignition current, car A, engine running slowly*

Note inequalities in time of contact

was cut off at *E*. It will be noted that these fluctuations in the charging current ceased at the time the ignition was cut off. Further study to determine the cause of these fluctuations showed that they occurred every time that the circuit was made through the primary coil of the ignition system. A magnification of these

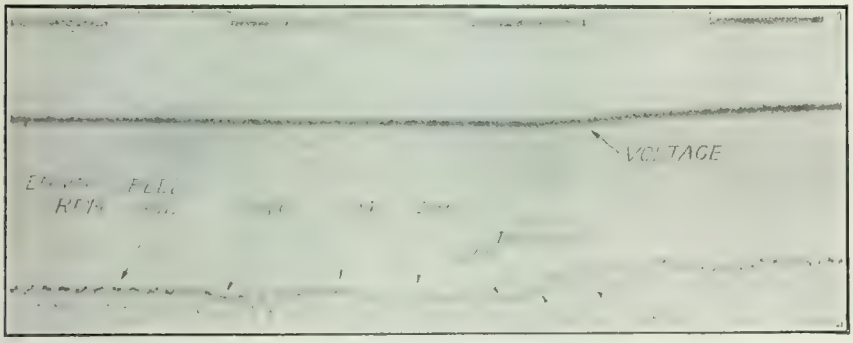

FIG. 27.—*Variation in charging current due to changing engine speed, car A*

The charging current changed to discharge current at 427 r p m of the crank shaft The speed at any time can be calculated by counting the interruptions with respect to the time scale

current fluctuations is shown in Fig. 26. Here it will be seen that the duration of time that the current flowed through the primary coil was appreciably longer in the case of the contacts marked 1 and 2 than for those marked 3 and 4. This inequality in the action of the contact points is probably due to inequalities in the cam which operates the contact points, or possibly to wear in the

distributor cam-shaft bearings. By comparing Fig. 26 with Fig. 10 it may be seen that the shape of the fluctuations in the current-charging curves are quite different although both had the same system for ignition. The difference is explained by the difference in the cams in the distributors.

Since the current passes through the primary of the ignition coil twice for each revolution of the crank shaft of the car A, it is possible to determine the exact speed of the engine by measuring the distance between any given number of these points in the charging curve with respect to the time scale. This is also possible for the car E, for which there are four contacts for each revolution. The exactness with which the speed of rotation may be measured by this method affords the possibility of determining small changes in the velocity of the flywheel system from one revolution to another, which can not be done by a tachometer.

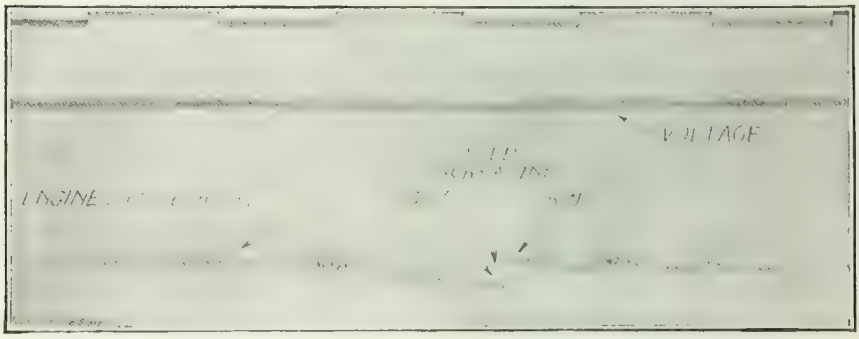

FIG. 28.—*Variations of charging rate at increasing speeds, car A*

The action of a limiting relay is shown at 907 r p m of the crank shaft

Figs. 27 and 28 show the characteristics of the charging current at varying speeds as observed on the car A. In Fig. 27 the speed was low, but in Fig. 28 it is much higher. The engine speed can be computed as described above. These curves are shown here to illustrate the applications of this method of measurement rather than to give quantitative measurements on this starter or system.

7. CONCLUSIONS AND SUMMARY

This investigation was begun to study the demands upon starting and lighting batteries in the operation of various types of automobiles. Illustrations of the oscillograms obtained for five of the cars measured have been included in this paper. These cars were loaned for these experiments by various individuals and

by the Motor Transport Corps of the Army. Dr. H. C. Dickinson, of this Bureau, and Majors M. O. Boone and G. H. Totten, of the Motor Transport Corps, gave valuable suggestions and cooperation.

The results obtained enable us to determine the instantaneous values of current and voltage when the starting system is in operation. The interpretation of the records has also indicated the possibility of using this method for the study of lubrication and engine problems relating to torque, speed, friction, ignition, compression, distributor action, etc. A large number of oscillograms illustrating these applications have been given. These are intended to be suggestive rather than quantitative measurements of performance, since the observations were all made on cars in running condition, without fixed orifices on the carbureter or constant speed control.

WASHINGTON, October 6, 1920.

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No. 187

A STUDY OF TEST METHODS FOR THE PURPOSE OF DEVELOPING STANDARD SPECIFICATIONS FOR PAPER BAGS FOR CEMENT AND LIME

BY

PAUL L. HOUSTON, Associate Physicist

Bureau of Standards

MARCH 21, 1921

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By Paul L. Houston

ABSTRACT

This technologic paper contains information relating to the methods of testing and the apparatus employed in determining the quality of paper bags for lime and cement. A brief description is given of the ordinary tests performed on paper, such as weight in pounds of the standard size ream 25×40—500, bursting strength in points, tensile strength in kilograms, stretch in centimeters, folding endurance in number of double folds, percentage of fiber composition, percentage of ash, and percentage of rosin sizing. A special test is developed for giving numerically the stresses and strains that the paper of these bags undergo in service. This test is called a resiliency or endurance test. A service test is also given to determine the breaking strength of the paper when the bags are filled and dropped. A comparison is made of the results of this test with the results of the above resiliency test. Another service test is developed for determinining the strength of the adhesives used in these bags. The results of all these tests are tabulated, and from these data the very best quality bags are chosen. In the conclusions, special consideration is given this choice of best bags in determining the characteristics of a good quality bag, and specifications are drawn up accordingly.

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I. INTRODUCTION

1. PURPOSE OF TESTS

The purpose of these tests is to aid the paper-bag manufacturers to meet the requirements of the lime and cement manufacturers in obtaining a suitable paper bag in which to ship their product. For a number of years the paper-bag manufacturers have attempted to manufacture such an article to substitute for the cotton bag

that had hitherto been used with great expense to the lime and cement industries because of the high price of cotton. The success of the paper-bag industry in this undertaking has been varied. A few of the consumers have given favorable reports on the use of their product, while others have claimed the bags fail in service. A number of consumers claim the glue gives way easily if the bags are held in storage any length of time. Since the bag manufacturers have been unable to derive any decided benefit from the above reports of the consumers, the present study was undertaken in order to develop specifications for these paper bags.

2. DISCUSSION OF MATERIALS AND APPARATUS USED

The materials used for this study were representative samples of paper bags obtained from a number of the leading bag manufacturers. The bags were given the following identification numbers: 14373, 14374, 14375, 14376, 14377, 14378, 14379, 14380, and 14381. The apparatus used was the Mullen paper tester, the paper scales, the 200 kg tensile-strength instrument, the 50 kg tensile-strength instrument, the folding endurance machine, microscope, glass slides, test tubes, extraction flasks, siphon cups, evaporating dishes and crucibles, all of which are used for ordinary physical and chemical tests. Descriptions and photographs of these instruments and apparatus may be obtained from circular of the Bureau of Standards No. 107, with the exception of the large 200 kg tensile-strength machine. The description and photograph of the tensile-strength apparatus in the above circular apply to the small 50 kg instrument. However, the 200 kg and the 50 kg instruments are practically identical in construction, the 200 kg machine being used for any material breaking at or under a load of 200 kg, the 50 kg machine for material breaking at or under a 50 kg load. In addition to the above paper-testing instruments a wooden platform with trapdoor and an ordinary foundry jolter were used for service tests. The wooden platform was built 3 feet high and allowed a bag filled with sand to be dropped through its trapdoor onto a solid wooden stand. The jolter consisted of a solid iron plate 3 inches thick which could be raised and lowered in rapid succession by means of air pressure. This machine, of course, was constructed with the necessary intake and outlet valves, air chamber, and piston to allow for the continuous jolting movement of the plate. For the purpose of developing a laboratory test that would give numerically the resiliency of bag

paper or the stress and strain that the paper of a bag undergoes in service and by which the results of the above service tests could be checked, the large 200 kg tensile-strength machine was adopted with the following additional attachments: A brass cylinder was constructed and fastened to the upright standard in such a way that it would revolve. The revolution of this cylinder was controlled by a small wire over a pulley from the weighted pendulum so that, as the pendulum moved from its vertical position due to the increasing loads, the cylinder revolved accordingly. This cylinder held a paper chart, and a glass recording pen was used to record both load and stretch. The pen was controlled by a small wire extending from the lower-jawed arm over a pulley in such a way that, as the lower jaw descended and the paper stretched under the load, the pen ascended and described a curve which registered stretch. It can thus be seen that the revolution of the cylinder caused the registering of load on the chart, while the ascension of the pen caused the registering of stretch. The result was a curve on the chart giving both the increasing load and increasing stretch until the paper broke. In order to read this curve it was necessary to draw up the chart so that load and stretch could be read at a glance. The method used was as follows: An ordinary blank piece of writing paper was used for this purpose and placed in position on the cylinder. A point on the paper was chosen as the zero point for load and stretch. Then the zero line for stretch was found by using a piece of steel between the jaws and by applying the various loads. Steel was used for this purpose because the chart was to be drawn up for a 70 kg load as a maximum, and steel would not stretch at this load. By this method the zero line for stretch was drawn with the recording pen from the zero point, and each kilogram load as indicated by the scale was pointed off as the load increased. At each kilogram point a line was drawn parallel to the base or zero line for load. This base line was found by operating the machine with nothing between the jaws and was drawn with the recording pen from the zero point. Each centimeter that the lower jaw descended from its starting position or zero point represented stretch and was pointed off accordingly. At the different centimeter points lines were drawn parallel to the zero line for stretch. In this way the chart was plotted to give accurately the stretch at different loads for any paper breaking under a 70 kg load.

3. SCOPE OF THE TESTS AND METHODS EMPLOYED

The object of these tests was, first, to discover all the physical and chemical properties of the paper of the bags; second, to check these results with the results of service tests on the bags themselves; third, to choose by the process of elimination the best bags of the series; and fourth, to develop specifications from this choice of best bags. For the ordinary physical and chemical tests the methods employed have been fully described in Circular of the Bureau of Standards No. 107, with the exception of the tensile strength and stretch tests. For these tests, the 200 kg instrument was used, and test samples were cut 2½ cm wide and long enough to allow 20 cm between jaws. In addition to the above tests a wet tensile test was conducted in the same manner as the dry tensile test described in the above Circular, with the following exceptions: The weight was removed from the pendulum of the 50 kg tensile-strength instrument, the machine was calibrated to find the corrections for the scale readings, and the paper was immersed for 10 minutes in water before testing. In both service tests bags were filled with 94 pounds of sand and then dropped or jolted until they broke open. In the jolter test the bags were jolted 1 inch high and 170 times per minute. An average of five tests was taken in each of these service tests to determine the strength of the bag. The method used for obtaining the resiliency of paper on the tensile-strength machine with its recording device was the following: In order to obtain curves or loops (as they should be called in this case) to show this resiliency or the endurance of paper under the repeated application of load, a load was taken in each case 10 per cent below the breaking point. This, of course, necessitated finding the tensile strength or breaking point for each grade of bag paper in both machine and cross direction. In order to do this an average of 10 tensile-strength tests was obtained. After obtaining in each case the desired load of 10 per cent below breaking point, each test sample was given this load, and then the load was released and the operation repeated until the test sample broke. All test samples for this work were cut 2½ cm wide and long enough to allow 20 cm of paper between jaws. Three test samples from each grade of paper, cut in both machine and cross direction, were used in order to obtain an average, and data were collected accordingly.

II. EXPERIMENTAL WORK

1. PHYSICAL AND CHEMICAL TESTS

Each sample of bag paper was given the ordinary physical and chemical tests, such as weight in pounds of the standard size ream 25×40—500, bursting strength in points, tensile strength in kilograms, stretch in centimeters, folding endurance in number of double folds, fiber composition in per cent, per cent of ash, and per cent of rosin sizing. From these tests the ratio in per cent of bursting strength to weight in pounds of the standard size ream, 25×40—500, and the breaking length in yards were obtained as the best means for comparing papers of the same grade. An explanation and description of this ratio and breaking length may be procured from Circular of the Bureau of Standards No. 107. The results of these tests are compiled in Tables 1, 2, and 4. In addition to the above tests the paper and bags were given the following special physical tests: The paper was tested for tensile strength in grams when it was wet, and was tested also for resiliency or endurance. The numerical results are shown in Tables 2, 3, and 4 and are presented graphically in Figs. 1 to 54. The bags were tested for breaking strength by determining the number of times each bag could be dropped before failure, as shown in Table 5, and for strength of adhesive by determining the length of time each bag could be jolted before failure, as shown in Table 6. The purposes of these special tests were the following: The wet tensile-strength test was made in order to indicate what the strength of the paper of the bags would be when they became wet in shipment. The stress-strain or resiliency test was made in order to obtain a series of loops for each test sample, giving first the stretch at applied load, second the stretch when load was released, third the increase in stretch for every time load was applied over the previous time, and fourth the number of times the paper could stand the strain of this load before it broke. The purpose of the drop test was to reproduce service conditions, because a bag is often dropped while being handled. On the other hand, the jolter test not only tested the strength of the adhesive used in the bags but also somewhat duplicated the treatment that a bag undergoes while being jolted and jarred in shipment from manufacturer to consumer. The following data were collected accordingly from all the above tests, and from these data it was not hard, by process of elimination, to choose the five bag papers which showed the best bursting strength, tensile strength, stretch,

folding endurance and resiliency, and which as bags showed the best breaking strength and possessed the strongest adhesive. At a glance, bag papers numbered 14373, 14374, 14378, 14379, and 14381 seem to be the strongest. In comparing the following results of the resiliency test with those results of the drop test, it will be noted that the above papers proved strongest in both cases. This proves that the resiliency test is a very good test for showing what the paper of a bag will do in actual service, and since it is strictly a laboratory test, it is highly to be recommended.

2. TABLES OF RESULTS

TABLE 1.—Results of Ordinary Physical and Chemical Tests on Cement and Lime Bag Paper

Bag-paper identification numbers	Weight of the standard size ream 25×40–500	Bursting strength in points	Ratio of bursting strength to weight of the standard size ream 25×40–500	Percentage fiber composition	Ash	Rosin sizing
	Pounds		Per cent		Per cent	Per cent
14373	150	134.0	89.3	100 jute and manila	4.10	3.40
14374	119	100.0	84.0	do	3.60	3.80
14375	143	108.0	75.6	95 jute and manila, 5 chemical wood	4.70	4.90
14376	127	96.0	75.6	100 chemical wood	3.30	3.30
14377	115	78.0	67.8	100 jute and manila	4.60	4.10
14378	176	132.0	75.0	50 jute and manila, 50 chemical wood	2.30	4.40
14379	142	143.0	107.0	75 jute and manila, 25 chemical wood	2.00	3.90
14380	172	159.0	92.4	100 chemical wood	1.27	.97
14381	181	155.0	85.7	do	.97	1.35

TABLE 2.—Results of Tensile-Strength Tests and Stress-Strain or Resiliency Tests on Cement and Lime Bag Paper

Bag-paper identification numbers	Weight of the standard size ream 25×40–500	Load at breaking point		Stretch at breaking point		Breaking length		Load 10 per cent below breaking point		Number of times load is applied	
		Machine direction	Cross direction	Machine direction	Cross direction	Machine direction	Cross direction	Machine direction	Cross direction	Machine direction	Cross direction
	Pounds	kg	kg	cm	cm	Yards	Yards	kg	kg		
14373	150	35.0	17.5	0.57	1.57	7130	3565	31.50	15.75	5.66	8.66
14374	119	28.0	16.5	.64	1.60	7160	4240	25.20	14.85	21.66	12.33
14375	143	27.0	11.0	.49	1.25	5770	2350	24.30	9.90	30.30	19.33
14376	127	26.5	13.0	.60	1.43	6380	3140	23.85	11.70	26.00	8.66
14377	115	21.5	8.8	.30	.70	5710	2340	19.35	7.92	11.00	11.66
14378	176	46.5	18.0	.54	1.34	5100	3120	41.85	16.20	49.66	4.66
14379	142	37.5	17.5	.64	1.06	8060	3770	33.75	15.75	12.00	9.66
14380	172	43.4	16.4	.45	1.12	7730	2910	39.06	14.76	1.00	2.00
14381	181	42.3	18.4	.50	1.14	7160	3110	38.07	16.56	59.00	9.66

TABLE 3.—Results of Stress-Strain or Resiliency Tests on Cement and Lime Bag Paper, Showing Different Stretches

	Loop 1	Loop 3	Loop 5	Loop 10	Loop 20	Loop 30	Loop 40	Loop 60
Bag identification No. 14373:								
Stretch at applied load—								
Machine direction....per cent..	1.68	2.01	2.30	.....	.....	.....	.....	.....
Cross direction....do....	5.66	5.77	.....	6.10	.....	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.25	.36	.40	.....	.....	.....	.....	.....
Cross direction....do....	.90	.91	1.03	1.05	.....	.....	.....	.....
Bag identification No. 14374:								
Stretch at applied load—								
Machine direction....per cent..	2.60	.....	.....	2.65	2.75	.....	2.95	3.15
Cross direction....do....	6.28	6.96	7.56	7.80	.....	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.30	.....	.42	.47	.51	.....	.54	.56
Cross direction....do....	.94	1.27	1.34	1.44	.....	.....	.....	.....
Bag identification No. 14375:								
Stretch at applied load—								
Machine direction....per cent..	1.18	.....	1.56	1.90	2.22	2.60	.....	.....
Cross direction....do....	3.85	5.13	.....	.....	5.40	.....	5.60	.....
Stretch when load is released—								
Machine direction....centimeters..	.27	.....	.36	.44	.50	.....	.....	.....
Cross direction....do....	.53	.86	.....	.....	.92	.....	1.00	.....
Bag identification No. 14376:								
Stretch at applied load—								
Machine direction....per cent..	1.75	.....	2.16	2.51	2.75	.....	.....	.....
Cross direction....do....	5.23	5.90	6.50	.....	6.60	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.31	.38	.44	.51	.60	.....	.....	.....
Cross direction....do....	.80	.99	.....	1.02	.....	.....	.....	.....
Bag identification No. 14377:								
Stretch at applied load—								
Machine direction....per cent..	1.16	.....	1.65	2.00	.....	.....	.....	.....
Cross direction....do....	2.90	3.75	.....	4.05	4.30	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.23	.....	.36	.43	.....	.....	.....	.....
Cross direction....do....	.40	.67	.....	.70	.75	.....	.....	.....
Bag identification No. 14378:								
Stretch at applied load—								
Machine direction....per cent..	1.68	.....	.....	1.90	.....	.....	.....	.....
Cross direction....do....	5.46	6.38	7.12	.....	.....	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.26	.....	.32	.40	.....	.....	.....	.....
Cross direction....do....	.82	1.06	1.20	.....	.....	.....	.....	.....
Bag identification No. 14379:								
Stretch at applied load—								
Machine direction....per cent..	2.31	.....	2.55	2.85	3.10	.....	.....	.....
Cross direction....do....	4.93	5.28	5.45	5.60	6.00	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.38	.50	.58	.64	.70	.....	.....	.....
Cross direction....do....	.80	.95	1.00	1.05	.....	.....	.....	.....
Bag identification No. 14380:								
Stretch at applied load—								
Machine direction....per cent..	2.10	.....	.....	.....	.....	.....	.....	.....
Cross direction....do....	3.13	3.60	.....	.....	.....	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.....	.....	.....	.....	.....	.....	.....	.....
Cross direction....do....	.50	.78	.....	.....	.....	.....	.....	.....
Bag identification No. 14381:								
Stretch at applied load—								
Machine direction....per cent..	1.53	.....	1.83	2.10	2.33	.....	.....	.....
Cross direction....do....	3.13	3.90	4.50	4.95	.....	.....	.....	.....
Stretch when load is released—								
Machine direction....centimeters..	.32	.....	.42	.49	.54	.....	.....	.....
Cross direction....do....	.44	.56	.66	.72	.....	.....	.....	.....

TABLE 4.—Results of Wet Tensile Tests and Folding Endurance Tests on Cement and Lime Bag Paper

Bag-paper identification numbers	Wet tensile strength		Breaking length		Folding endurance in number of double folds	
	Machine direction	Cross direction	Machine direction	Cross direction	Machine direction	Cross direction
	kg	kg	Yards	Yards		
14373	1.38	1.40	476	482	10 000+	10 000+
14374	2.36	1.71	1026	746	10 000+	10 000+
14375	1.43	1.29	519	468	10 000+	1 998
14376	2.05	1.68	836	685	6 220	4 260
14377	1.17	.92	525	418	10 000+	5 011
14378	3.15	1.35	926	396	10 000+	10 000+
14379	1.31	.71	349	262	10 000+	10 000+
14380	.80	.30	243	90	7 244	10 000+
14381	1.26	.69	362	199	10 000+	10 000+

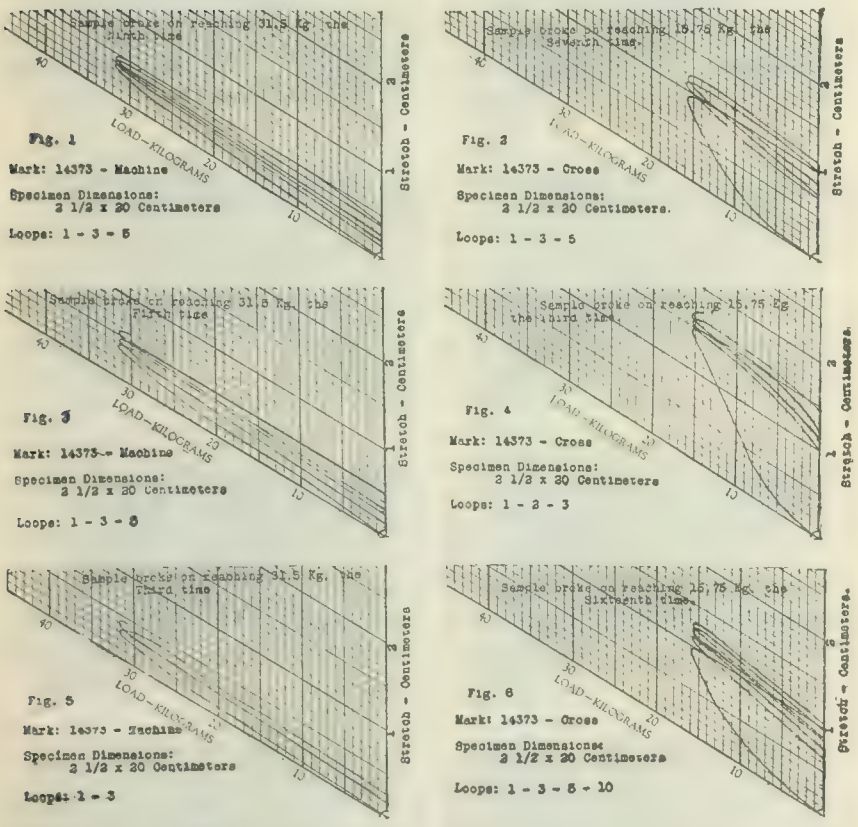
TABLE 5.—Results of Drop Service Tests on Cement and Lime Bag Paper

Bag-paper identification numbers	Position of bag when dropped	Number of drops before failure	Extent of failure
14373	On side and seam up	2	Paper failed at seam.
14374	do	3	Do.
14375	do	1	Paper and glue failed at seam.
14376	do	1	Paper failed across edge, and glue failed at end.
14377	do	1	Paper and glue failed at seam, and paper failed across bottom side.
14378	do	4	Paper failed at seam.
14379	do	4	Do.
14380	do	1	Paper failed across end, from side to side, and glue failed at end.
14381	do	3	Glue failed at seam and end.

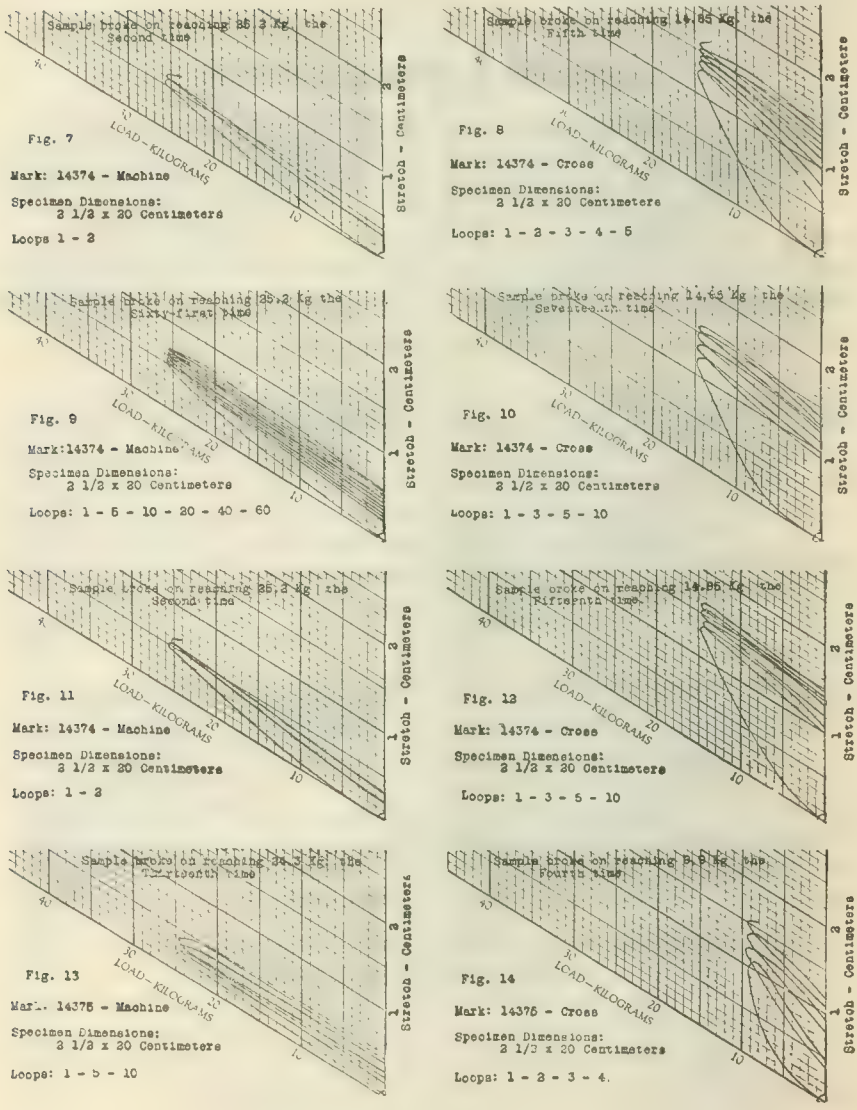
TABLE 6.—Results of Jolter Service Tests on Cement and Lime Bag Paper

Bag-paper identification numbers	Position of bag when jolted	Minutes jolted before failure	Extent of failure
14373	On side and seam up	24	Glue failed at end.
14374	do	60	No failure.
14375	do	1	Glue failed at end.
14376	do	18	Do.
14377	do	6	Do.
14378	do	40	Do.
14379	do	30	Do.
14380	do	2	Do.
14381	do	3	Do.

3. DIAGRAMS ILLUSTRATING RESULTS


Sample broke on reaching 31.5 Kg. the Ninth time
Fig. 1
Mark: 14373 - Machine
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3 - 5
Sample broke on reaching 15.75 Kg. the Seventh time.
Fig. 2
Mark: 14373 - Cross
Specimen Dimensions:
2 1/2 x 20 Centimeters.
Loops: 1 - 3 - 5
Sample broke on reaching 31.5 Kg. the Fifth time
Fig. 3
Mark: 14373 - Machine
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3 - 5
Sample broke on reaching 15.75 Kg. the third time.
Fig. 4
Mark: 14373 - Cross
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 2 - 3
Sample broke on reaching 31.5 Kg. the Third time
Fig. 5
Mark: 14373 - Machine
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3
Sample broke on reaching 15.75 Kg. the Sixteenth time.
Fig. 6
Mark: 14373 - Cross
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3 - 5 - 10
LOAD - KILOGRAMS
Stretch - Centimeters

FIGS. 1 to 6.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*


Sample broke on reaching 35.3 Kg. the Second time
Fig. 7
Mark: 14374 - Machine
Specimen Dimensions: 2 1/2 x 20 Centimeters
Loops 1 - 2
Sample broke on reaching 14.65 Kg. the Fifth time
Fig. 8
Mark: 14374 - Cross
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 2 - 3 - 4 - 5
Sample broke on reaching 35.2 Kg the Sixty-first time
Fig. 9
Mark:14374 - Machine
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 5 - 10 - 20 - 40 - 60
Sample broke on reaching 14.65 Kg the Seventeenth time
Fig. 10
Mark: 14374 - Cross
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 3 - 5 - 10
Sample broke on reaching 35.3 Kg the Second time
Fig. 11
Mark: 14374 - Machine
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 2
Sample broke on reaching 14.95 Kg the Fifteenth time
Fig. 12
Mark: 14374 - Cross
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 3 - 5 - 10
Sample broke on reaching 34.7 Kg. the Thirteenth time
Fig. 13
Mark: 14375 - Machine
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 5 - 10
Sample broke on reaching 9.9 Kg the Fourth time
Fig. 14
Mark: 14375 - Cross
Specimen Dimensions: 2 1/2 x 30 Centimeters
Loops: 1 - 2 - 3 - 4
LOAD - KILOGRAMS
Stretch - Centimeters

FIGS. 7 to 14.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*

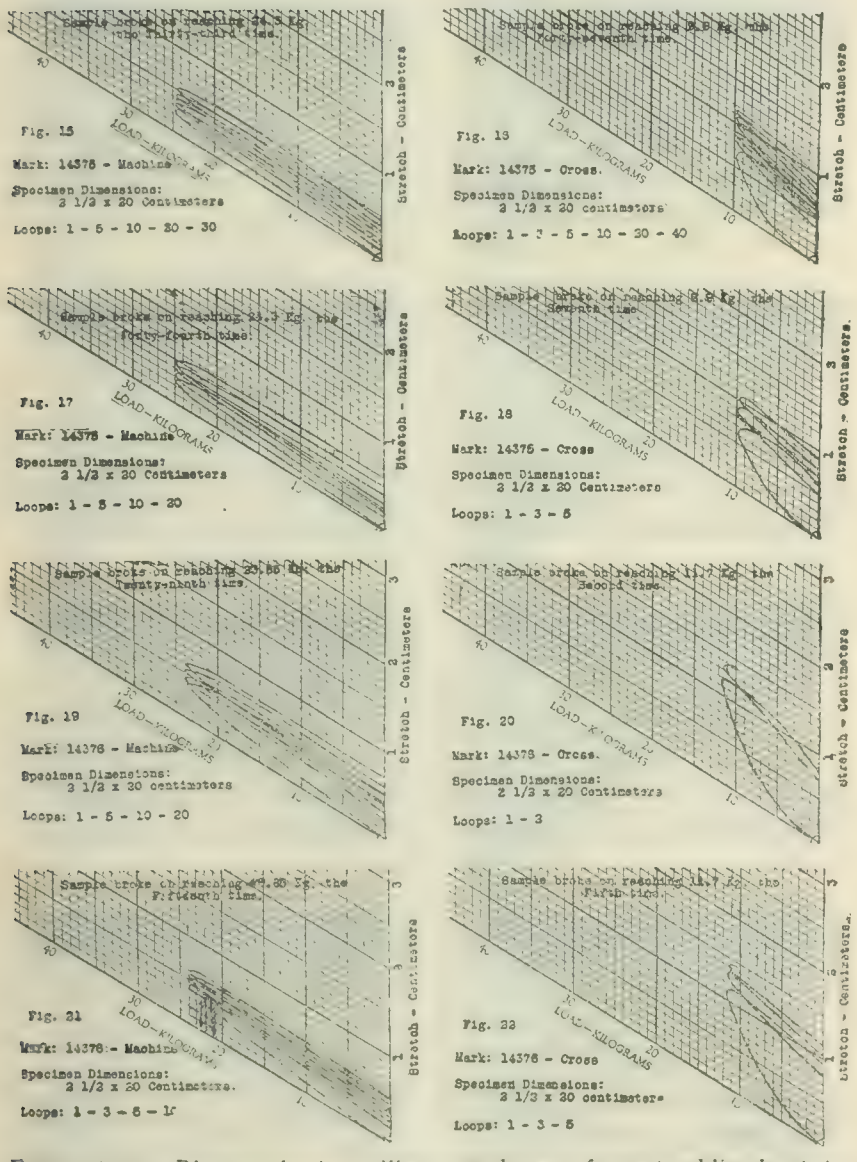
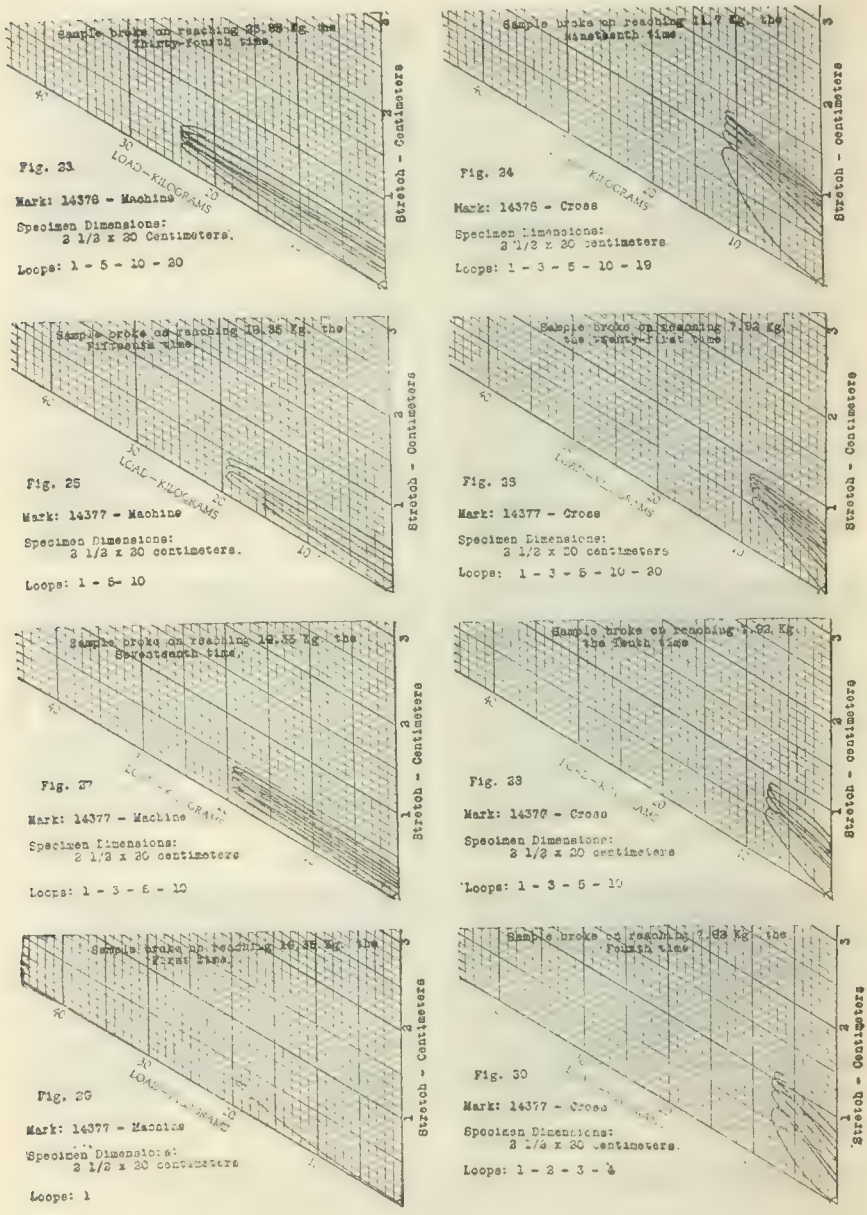


Fig. 15
Mark: 14376 - Machine
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 5 - 10 - 20 - 30
Fig. 16
Mark: 14375 - Cross.
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10 - 30 - 40
Fig. 17
Mark: 14376 - Machine
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 5 - 10 - 20
Fig. 18
Mark: 14375 - Cross
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3 - 5
Fig. 19
Mark: 14376 - Machine
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 5 - 10 - 20
Fig. 20
Mark: 14375 - Cross.
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3
Fig. 21
Mark: 14376 - Machine
Specimen Dimensions:
2 1/2 x 20 Centimeters.
Loops: 1 - 3 - 5 - 10
Fig. 22
Mark: 14376 - Cross
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 3 - 5
LOAD - KILOGRAMS
Stretch - Centimeters

FIGS. 15 to 22.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*


Sample broke on reaching 25.89 Kg. the thirty-fourth time.
Fig. 23
Mark: 14376 - Machine
Specimen Dimensions: 2 1/2 x 20 Centimeters.
Loops: 1 - 5 - 10 - 20
Sample broke on reaching 11.7 Kg. the nineteenth time.
Fig. 24
Mark: 14376 - Cross
Specimen Dimensions: 2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10 - 19
Sample broke on reaching 19.35 Kg. the fifteenth time.
Fig. 25
Mark: 14377 - Machine
Specimen Dimensions: 2 1/2 x 20 centimeters.
Loops: 1 - 5 - 10
Sample broke on reaching 7.02 Kg. the twenty-first time
Fig. 26
Mark: 14377 - Cross
Specimen Dimensions: 2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10 - 20
Sample broke on reaching 12.35 Kg. the Seventeenth time.
Fig. 27
Mark: 14377 - Machine
Specimen Dimensions: 2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10
Sample broke on reaching 4.92 Kg. the Tenth time
Fig. 28
Mark: 14376 - Cross
Specimen Dimensions: 2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10
Sample broke on reaching 16.35 Kg. the First time.
Fig. 29
Mark: 14377 - Machine
Specimen Dimensions: 2 1/2 x 20 centimeters
Loops: 1
Sample broke on reaching 7.03 Kg. the Fourth time.
Fig. 30
Mark: 14377 - Cross
Specimen Dimensions: 2 1/2 x 20 centimeters.
Loops: 1 - 2 - 3 - 4
Stretch - Centimeters
LOAD - KILOGRAMS

FIGS. 23 to 30.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*

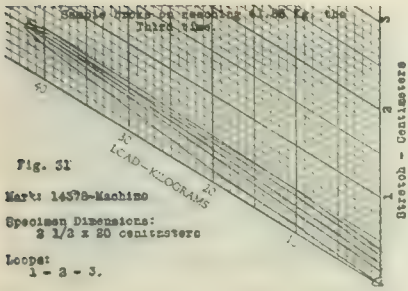


Fig. 31
Mark: 14378-Machine
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops:
1 - 2 - 3.

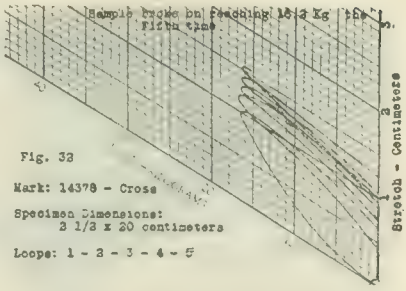


Fig. 32
Mark: 14378 - Cross
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 2 - 3 - 4 - 5

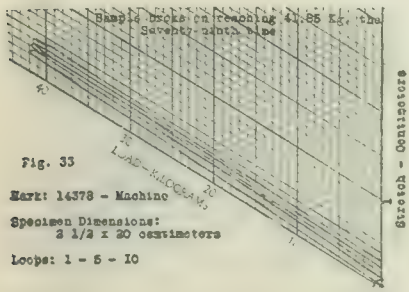


Fig. 33
Mark: 14378 - Machine
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 5 - 10

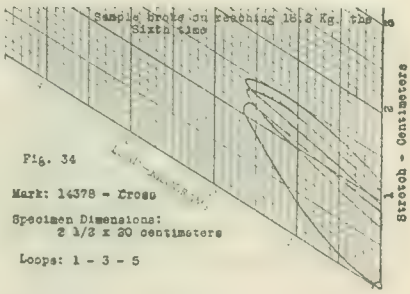


Fig. 34
Mark: 14378 - Cross
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 3 - 5

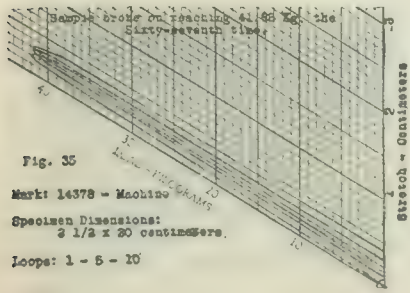


Fig. 35
Mark: 14378 - Machine
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 5 - 10

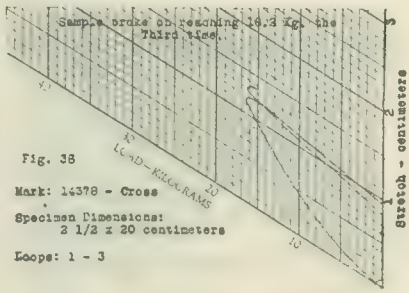


Fig. 36
Mark: 14378 - Cross
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 3

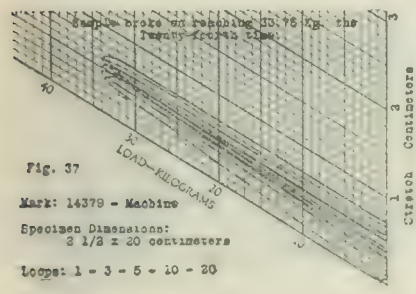


Fig. 37
Mark: 14379 - Machine
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10 - 20

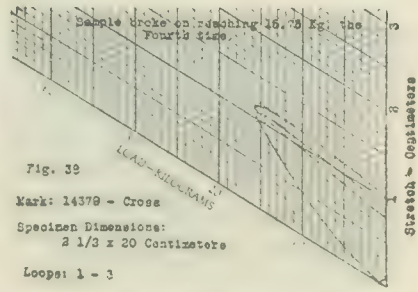
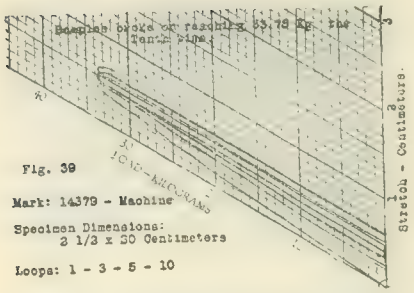
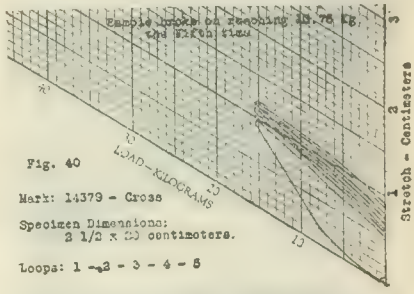
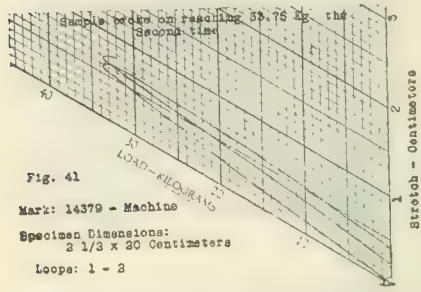


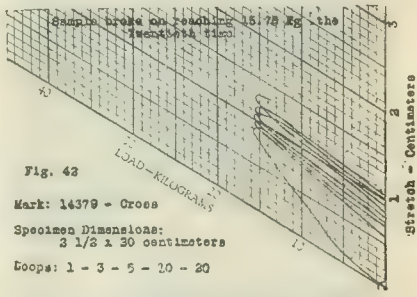
Fig. 38
Mark: 14379 - Cross
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 3

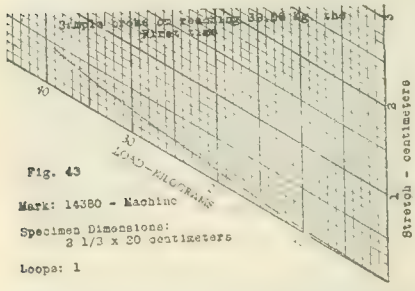
FIGS. 31 to 38.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*

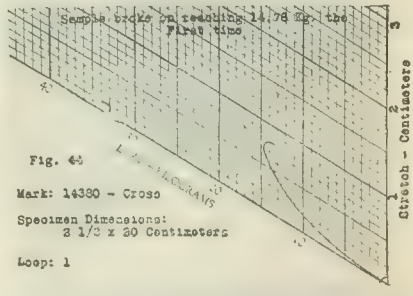

Sample broke on reaching 33.75 Kg. the Tenth time.
Stretch - Centimeters
LOAD - KILOGRAMS
Fig. 39
Mark: 14379 - Machine
Specimen Dimensions:
2 1/3 x 20 Centimeters
Loops: 1 - 3 - 5 - 10

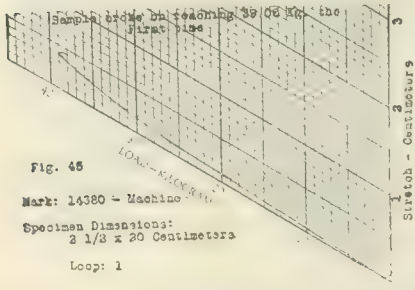

Sample broke on reaching 33.75 Kg. the Fifth time
Stretch - Centimeters
LOAD - KILOGRAMS
Fig. 40
Mark: 14379 - Cross
Specimen Dimensions:
2 1/2 x 20 centimeters.
Loops: 1 - 2 - 3 - 4 - 5

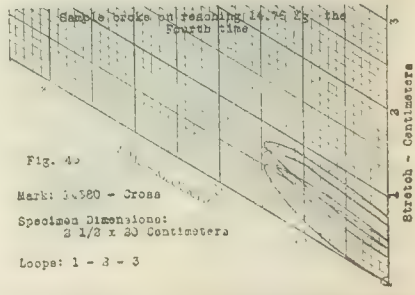

Sample broke on reading 33.75 Kg. the Second time
Stretch - Centimeters
LOAD - KILOGRAMS
Fig. 41
Mark: 14379 - Machine
Specimen Dimensions:
2 1/3 x 20 Centimeters
Loops: 1 - 2


Sample broke on reading 33.75 Kg. the Seventh time
Stretch - Centimeters
LOAD - KILOGRAMS
Fig. 42
Mark: 14379 - Cross
Specimen Dimensions:
2 1/2 x 20 centimeters
Loops: 1 - 3 - 5 - 10 - 20

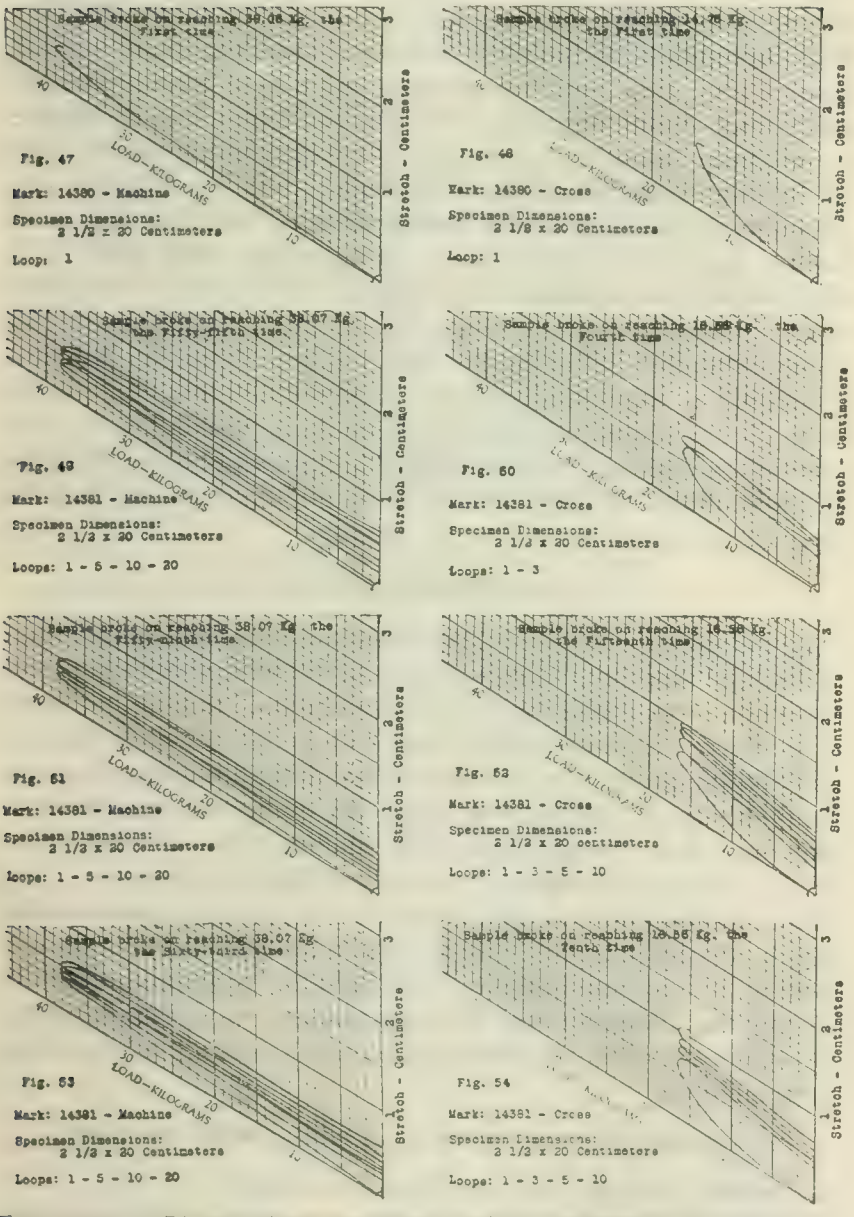







Sample broke on reading 14.76 Kg. the Fourth time
Stretch - Centimeters
Fig. 46
Mark: 14380 - Cross
Specimen Dimensions:
2 1/2 x 20 Centimeters
Loops: 1 - 2 - 3

FIGS. 39 to 46.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*



FIGS. 47 to 54.—*Diagrams showing resiliency or endurance of cement and lime bag paper in the machine and cross directions*

III. CONCLUSIONS

Conclusions to be drawn from the above results of tests, taking into special consideration the various tests on the chosen five best bags, are that a satisfactory cement or lime bag paper should have a very good bursting strength and a high tensile strength and stretch in both directions. It must also have a high endurance or resiliency, which is determined by the simple drop service test or by the better stress-strain laboratory test. A suitable bag paper must also show a good tensile strength when wet and must have a high folding endurance. As the above data show, it is not necessary to manufacture a bag out of very heavy paper in order to obtain a bag of the very best quality. For instance, bag paper numbered 14374 weighs only 119 pounds to the standard size ream 25×40—500, and yet is one of the five best papers tested. The fiber composition of a good bag paper should be not less than 50 per cent strong manila and jute and the remainder chemical wood. The ash should not be over 3 per cent and the rosin sizing should be at least 3.5 per cent.

IV. SPECIFICATION

In writing up the specification for cement and lime paper bags it does not seem wise to recommend any specific weight of paper, since this study shows that weight alone should not be considered in choosing the best-quality bags. It is for the consumer to decide what weight bag he wishes to use and for the manufacturer to decide what weight bag paper he wishes to manufacture. However, it is thought that both manufacturer and consumer would prefer a light-weight paper for reasons of economy. For these reasons weight is not given in the following specification, but in all other respects it is thought that this specification will meet all requirements and can be adhered to by the manufacurers without any difficulty, and that bags made of paper that conforms to this specification will prove satisfactory to the cement and lime manufacturers.

SPECIFICATIONS FOR PAPER BAGS FOR CEMENT AND LIME

Bursting strength: Not less than 100 points.

Ratio bursting strength to weight in pounds of the standard-size ream, 25×40—500: Not less than .75 per cent.

Fiber composition: Not less than 50 per cent manila and jute; the remainder as chemical wood.

Ash: Not over 3 per cent.

Rosin: Not less than 3.5 per cent.

Tensile strength (test sample 2½ cm wide and 20 cm between jaws): Not less than 28 kg (machine direction), 16 kg (cross direction).

Breaking length: Not less than 7000 yards (machine direction), 3500 yards (cross direction).

Wet tensile strength (test sample 15 mm wide and 90 mm between jaws): Not less than 2000 g (machine direction), 1300 g (cross direction).

Wet breaking length: Not less than 800 yards (machine direction), 400 yards (cross direction).

Folding endurance (test sample 15 mm wide and 90 mm between jaws): Not less than 10 000 double folds (machine direction), 10 000 double folds (cross direction).

Resiliency strength (test sample 2½ cm wide and 20 cm between jaws): Not less than 20 times (machine direction), 9 times (cross direction).

Breaking strength of bags: Not less than 3 times.

Adhesive strength: Not less than 30 minutes.

WASHINGTON, September 20, 1920.

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BUREAU OF STANDARDS

S. W. STRATTON DIRECTOR

No. 188

SOME PROPERTIES OF WHITE METAL BEARING ALLOYS AT ELEVATED TEMPERATURES

BY

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Bureau of Standards

APRIL 5, 1921

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SOME PROPERTIES OF WHITE METAL BEARING ALLOYS AT ELEVATED TEMPERATURES

By John R. Freeman, Jr., and R. W. Woodward

ABSTRACT

An apparatus is described for determining the yield point and ultimate strength of white metal bearing alloys at temperatures up to 100° C. A new design of heating apparatus is described for determining the Brinell hardness of metals at temperatures up to 100° C.

The results of compression tests and Brinell hardness tests at temperatures up to 100° C are given for five typical white metal bearing alloys, including three tin-base alloys, one lead-base alloy, and one intermediate alloy, which show that the tin-base alloys maintain their properties better at elevated temperatures than the lead-containing alloys.

Results of tests are given which indicate that up to 5 per cent of lead in a high-grade babbitt does not affect the yield point or ultimate strength at 25 or 75° C.

The yield point of tin-base alloys is not affected by heating for six weeks at about 100° C, but the yield point is lowered in the lead-base alloy by heating for two weeks at about 100° C.

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I. INTRODUCTION

The mechanical properties of white metal bearing alloys have been the subject of several investigations with the particular object of establishing the relations between the properties obtained in laboratory tests and the ultimate test of service. Practically all of these tests reported in the literature on white metal bearing alloys were conducted at room temperature, the Brinell hardness alone[1] having been determined for a few alloys under other conditions of temperature. While we therefore have considerable knowledge of the mechanical properties of these alloys at ordinary temperatures, our knowledge of their properties at elevated temperatures is very limited.

The importance of knowing the properties of bearing alloys at elevated temperatures is readily appreciated when one considers that the oil temperature in the crank case of an automobile engine may often reach 60° C, and that bearing temperatures of 100° C and higher have been measured in similar engines.

It is the purpose of this paper to present the design of new apparatus and the results of tests to determine the mechanical properties in compression and the Brinell hardness of some representative white metal bearing alloys at elevated temperatures.

The nonferrous metals division of the Society of Automotive Engineers is proposing as standard white metal bearing alloys the four compositions given in Table 1.

TABLE 1.—Specifications for White Metal Bearing Alloys Proposed by the Society of Automotive Engineers

Components	S. A. E. No. 10	S. A. E. No. 11	S. A. E. No. 12	S. A. E. No. 13
	Per cent	Per cent	Per cent	Per cent
Tin	90–92	86–89	Remainder	4.50– 5.50
Copper	4–5	5–6.50	2.25– 3.75	<0.50
Antimony	4–5	6–7.50	9.50–11.50	9.25–10 75
Lead	<0.35	<0.35	24–26	84–86

These four alloys and A. S. T. M. alloy No. 2[2] were selected as of representative composition suitable for this investigation. It will be noticed that A. S. T. M. alloy No. 2 (Table 2) is very similar

[1] Jesse L. Jones, Babbitt and babbitted bearings, A. I. M. M. E. Trans., **60**, p. 458; 1919.

[2] "Tentative specifications for white metal bearing alloys," Proc. A. S. T. M., **19**, pt. 1, p. 469.

to the S. A. E. alloy No. 11. The properties of these two alloys are compared here, as the A. S. T. M. alloy was considered by the S. A. E. committee as a little hard.

The results of chemical analyses[3] of the five alloys studied are given in Table 2.

TABLE 2.—Percentage Composition of Alloys Studied

Components	Bearing metal No. 1: S. A. E. No. 10; A. S. T. M. No. 1	Bearing metal No. 2: A. S. T. M. No. 2	Bearing metal No. 3: S. A. E. No. 11	Bearing metal No. 4: S. A. E. No. 12	Bearing metal No. 5: S. A. E. No. 13; A. S. T. M. No. 9
	Per cent	Per cent	Per cent	Per cent	Per cent
Copper	4.56	3.51	5.65	2.90	.....
Antimony	4.52	7.57	6.90	10.50	10.03
Tin	Remainder	Remainder	Remainder	Remainder	Remainder
Lead	None	None	0.09	25.05	84.95
Iron	<0.05	<0.05	< .05	< .05	< .05

II. PREPARATION OF ALLOYS

1. METALS USED

Pure Banka tin and the best grade of "Star" antimony were used. Neither these metals nor the copper were analyzed, as the freedom of the alloys from impurities is proof of the purity of the metals used.

Analysis of the commercially pure lead used showed the following: Lead, 99.94 per cent; copper, 0.03 per cent; antimony, <0.03 per cent.

2. ALLOYING PROCEDURE

Alloys Nos. 1, 2, 3, and 4 were prepared by first melting the tin in a plumbago crucible in a gas furnace and then adding the requisite amounts of a 50 per cent Sn.-50 per cent Cu hardener and metallic antimony. After addition of the alloying elements the temperature of the bath was carried up to the melting point of antimony and stirred to insure a homogeneous alloy. The temperature of the melt was then allowed to drop to about 500° C with continual stirring of the metal, which was then poured into a cast-iron mold. The surface of the bath was always kept covered with charcoal to prevent excessive oxidation. The temperature was measured by means of a specially calibrated chromel-alumel thermocouple connected to a portable potentiometer.

[3] J. A. Scherrer, of the Bureau of Standards, made all chemical analyses reported in this paper.

The lead-base alloy No. 5 was similarly prepared, but in this case the lead was first melted and then the metallic tin and antimony were added.

The alloys were made up to meet the mean composition of the specifications, and the resultant compositions given indicate how close a desired composition may be obtained with the careful laboratory methods used.

III. PREPARATION OF TEST SPECIMENS

The compression test specimens used were small cylinders 1½ inches long by about ½ inch diameter (1.5 by 0.514 inch), this ratio of length to diameter being within the limits recommended by the A. S. T. M., and the cross section for these experiments being easily tested in a 10 000-pound testing machine. These specimens were turned in a lathe, with a hollow mill, from castings 2 inches long by ¾ inch diameter which were made by pouring the metal from the desired temperature into a split steel mold of the above dimensions.

The samples for Brinell testing were similar to those used by Lynch,[4] the metal being poured into an open steel mold 2 inches in diameter by ⅝ inch deep, but in this case the mold was not previously heated before pouring, always being at room temperature when the metal was first poured. Before making the impressions, the faces of the casting were turned off and the test then made on the bottom face. Three impressions were made on each casting at equidistant points on a circle one-half the radial distance from the center. The average of these three readings was taken as the Brinell hardness under the given conditions.

IV. APPARATUS USED FOR TESTING

1. COMPRESSION TESTS

The cylinders were compressed in a standard Reihle 10 000-pound testing machine.

The deformation per unit load was measured by a specially designed compressometer. A copy of a photograph of this instrument mounted on a specimen is shown in Fig. 1. The frame and uprights are made of aluminum. They are held to the specimen by three small steel screws, having conical points, set radially in the same plane, and spaced equidistantly around

[4] T. D. Lynch, "Study of bearing metals and methods of testing," A. S. T. M., 13, p. 699; 1913.

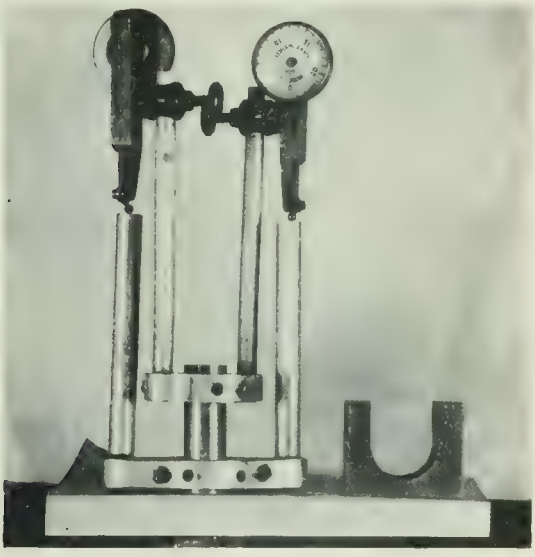

FIG. 1.—*Compressometer mounted on specimen*

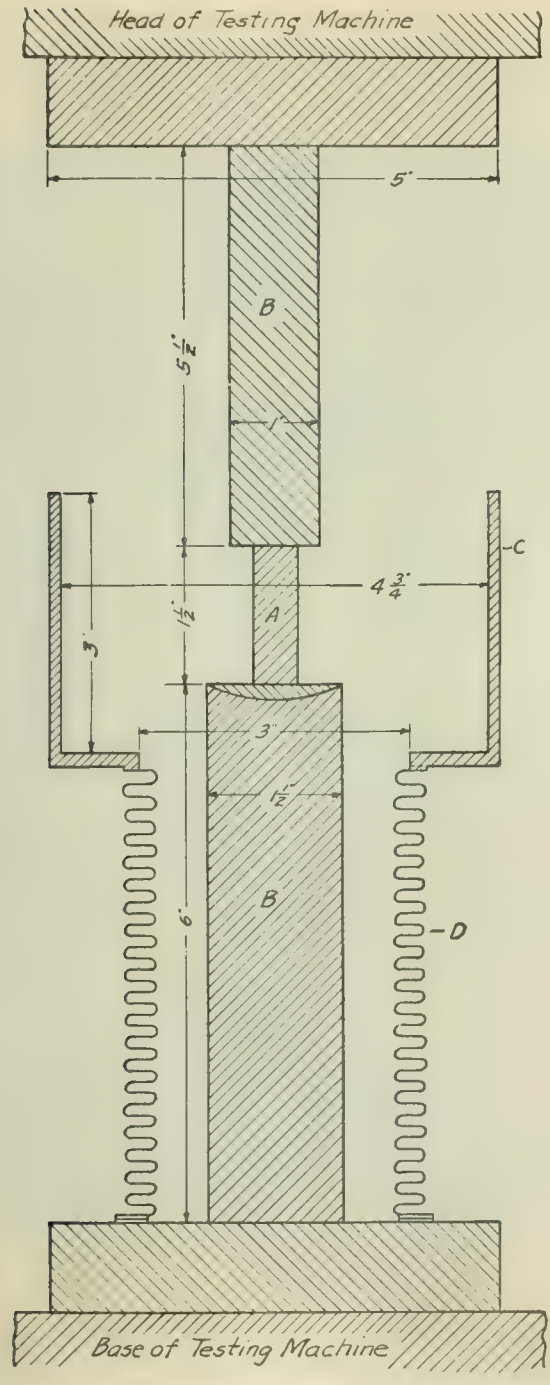

FIG. 2.—*Assembly drawing of a section of heating bath, showing original dimensions*

the specimen, as may be seen in the photograph. The small U-shaped block is a gage used for spacing the frames at the proper distance on the specimen. The gage length or the distance between the planes of the screws is 1 inch.

The "Last Word" dials used read to approximately thousandths of an inch (one division on dial = 0.00087 in.), and ten thousandths are readily estimated.

The assembly and important dimensions of the bath used for heating the specimen during tests are shown in Fig. 2, which is a section through the center. The specimen *A* is compressed between the steel posts *B B*. During test the specimen with compressometer attached is immersed in a heated liquid (glycerin was found very satisfactory) held in the container *C*. *D* is a Silphon diaphragm. This collapses like an accordion, permitting the top of the container *C* to drop below the level of the base of the specimen. This is a particularly convenient method for lowering the bath to place a specimen in position for testing, especially as it eliminates the need for any packed joints. A photograph of the entire apparatus with a specimen in position for testing is shown in Fig. 4.

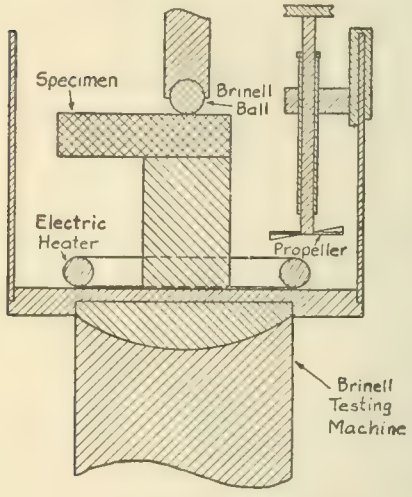

FIG. 3.—*Apparatus for Brinell hardness testing at elevated temperatures*

The bath is heated with a small size Hot-Point electric heater immersed in the glycerin. The glycerin was forced in a continuous stream over the heater by a small electric motor-driven propeller. This continuous stirring of the glycerin and the ready control of the heating current with a variable resistance provided excellent control of the temperature of the specimen during the test. In all cases the temperature of the bath and consequently the specimen did not vary by more than 2° C during a test.

2. BRINELL HARDNESS TESTS

The Brinell hardness tests were made with a standard Brinell machine using a 500 kg load on a 10 mm ball applied for 30 seconds. For the elevated temperature tests the apparatus shown in Fig. 3 was used. It is simply a suitable container for the heat-

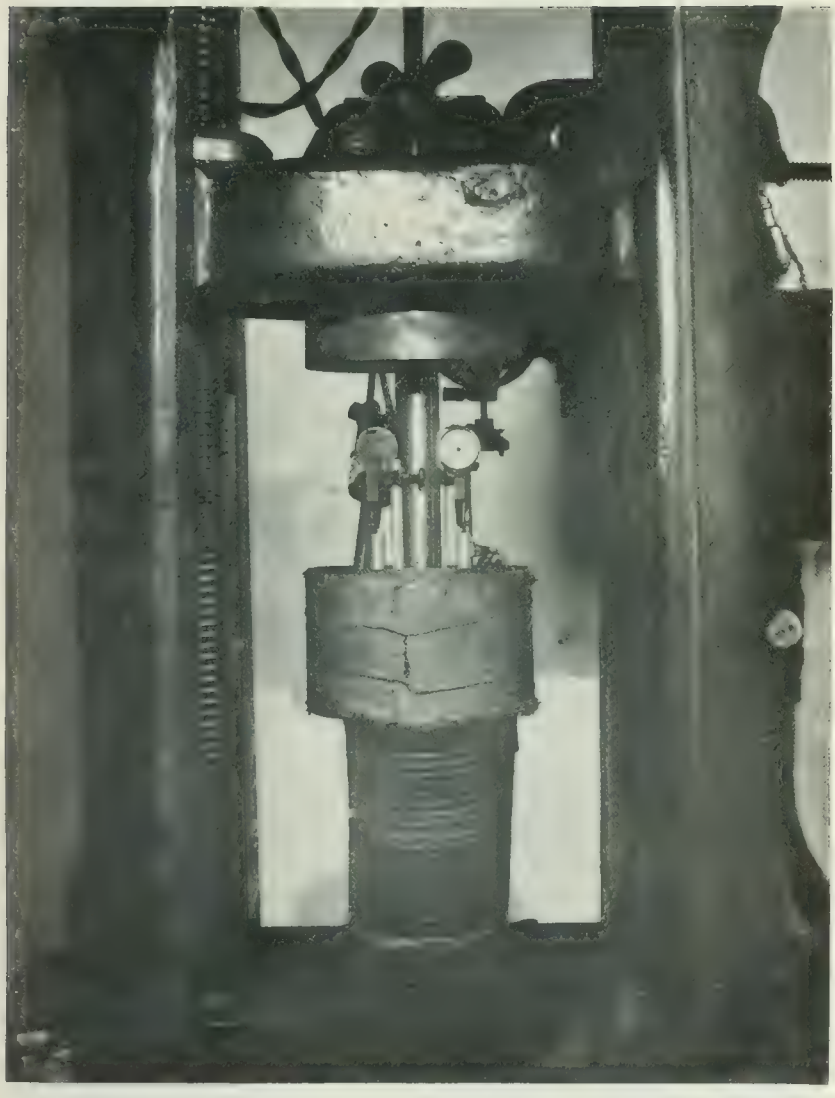

FIG. 4.—*Specimen with compressometer and heating bath assembled in testing machine*

ing liquid (glycerin was used) with a base made to fit on the spherical seat of the Brinell machine and a post on the inside to support the specimen away from the bottom and permit good circulation of the liquid around it. The bath was stirred with a small motor-driven propeller and was heated by a small resistor placed on the bottom of the container. During test the entire specimen was submerged, the Brinell ball also being completely immersed. Sufficient time was always allowed for the specimen to reach the temperature of the bath, this having been previously determined by inserting a thermocouple in a specimen and noting the time elapsed between the placing of the specimen in the bath and when the center reached the temperature of the bath.

V. PRELIMINARY TESTS

It is well known that the pouring temperature of a bearing metal, all other conditions being constant, has a marked influence on the mechanical properties. In view of this fact, preliminary to any test at elevated temperatures, the effect of pouring temperature on the compressive strength at room temperature was determined for alloys Nos. 1, 3, 4, and 5. The results obtained from compression tests are given in Table 3.

TABLE 3.—Effect of Pouring Temperatures on Yield Point and Ultimate Strength in Compression of Various Alloys

Alloy No.	Pouring temperature	Yield point	Ultimate strength
	°C	Lbs./in.²	Lbs./in.²
1	400	3750	12 940
	446	4000	12 855
	495	3500	13 500
3	390	3500	15 830
	445	4250	16 435
	500	4000	15 830
4	300	5000	14 015
	350	4250	13 685
	400	4750	13 635
5	300	3250	13 840
	356	3750	15 020
	404	3250	15 245

In results reported in this paper the yield point was adopted arbitrarily as at ⅛ of 1 per cent reduction of the gage length. The ultimate strength was arbitrarily chosen as the unit load necessary to produce a deformation of 25 per cent of the original

length of the test specimen. The reasons for selecting these values will be discussed later.

From a comparison of the results of Table 3 and the pouring temperatures suggested in the tentative specifications of the A. S. T. M.[5] the following temperatures were used in casting test specimens for all further tests.

	Pouring temperature, ° C
No. 1	440
No. 2	440
No. 3	440
No. 4	345
No. 5	325

VI. ELEVATED-TEMPERATURE COMPRESSION TESTS

Stress deformation curves were taken on all five alloys at room temperature (20–30° C), 50, 75, and 100° C. At least two specimens were tested under each condition to provide a check.

Representative stress-strain curves at the four temperatures of each alloy except No. 2 are given in Fig. 5. These show the type of stress-deformation curve obtained with the apparatus described in this paper and also show very clearly the marked change in the compressive strength of the alloys with increasing temperatures.

On the plot a "dial unit" is equivalent to 0.00087 inch and is the algebraic mean of the total deformation shown by the individual dials for any given load.

A study of the curves shows that it is practically impossible to pick out a limit of proportionality as ordinarily determined by noting the departure of the stress-deformation curve from a straight line, and, further, we know that the finer the measurement the lower will be this point. An arbitrary yield point was therefore determined upon. After comparing the yield points indicated by several values of percentage reduction of gage length the value of $\frac{1}{8}$ of 1 per cent of the gage length (0.00125 inch) was adopted for purposes of comparison, as it generally seems to coincide with the first marked yielding of the specimens tested. This value 0.00125 inch) is practically equivalent to 1.5 division on the dial or "dial units" used on plot (Fig. 5).

When soft metals of this type and size of test specimens are compressed they do not eventually shear but continue to flatten

[5] See footnote 2.

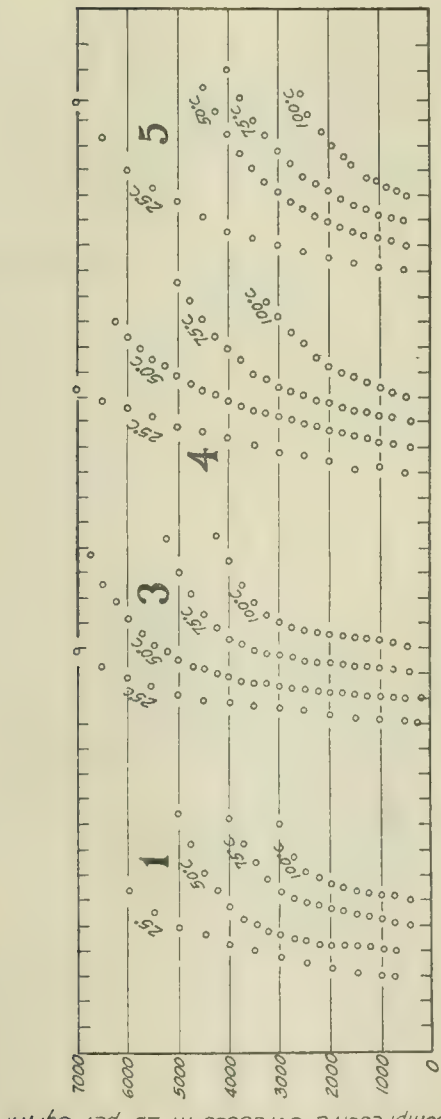

FIG. 5.—*Representative stress-strain curves of alloys Nos. 1, 3, 4, and 5 (see Table 2) at 25, 50, 75, and 100° C*

out with increasing loads, so it is necessary to adopt some arbitrary values for ultimate strength which will at least be comparable among themselves. A reduction of 25 per cent of length was chosen in this investigation as at this value in all cases the load had become nearly constant for increasing deformation. In the case of high lead alloys the load generally reached a maximum value and then fell off before the 25 per cent reduction was reached. In these cases the maximum load was recorded. The values of yield point and ultimate strength thus obtained are given in Table 4.

TABLE 4.—Yield Point and Ultimate Strength in Compression at Elevated Temperatures

Alloy No.	Property	Values in pounds per square inch at—			
		25° C	50° C	75° C	100° C
1	Yield point	4 400	3 800	3 150	2650
	Ultimate strength	12 850	10 400	8 450	6950
2	Yield point	6 250	4 850	4 000	2850
	Ultimate strength	15 175	11 850	9 400	6825
3	Yield point	5 750	5 000	4 250	3350
	Ultimate strength	16 425	12 175	10 100	7725
4	Yield point	4 700	3 650	2 900	2150
	Ultimate strength	13 685	10 035	7 845	6045
5	Yield point	3 750	2 650	2 250	1550
	Ultimate strength	15 020	11 275	7 920	4770

VII. ELEVATED TEMPERATURE BRINELL TESTS

The Brinell hardness of alloys Nos. 1, 3, 4, and 5 was determined at room temperature, 50, 75, and 100° C. The values obtained are given in Table 5.

TABLE 5.—Brinell Hardness at Elevated Temperatures

Alloy No.	Brinell hardness numeral at—			
	25° C	50° C	75° C	100° C
1	[a] 17.2 (28.6)	13.8	11.1	8.2 (12.8)
3	22.3	18.2	14.8	11.3
4	22.4	15.8	11.3	7.5
5	19.7 (19.5)	16.8	11.4	8.2 (8.6)
2	22.3 (28.3)			

[a] A. S. T. M specifications (see footnote 2) give the values shown in parentheses.

VIII. DISCUSSION OF RESULTS

1. COMPRESSION TESTS

For greater convenience of comparison the yield points of the four alloys are plotted against temperature in Fig. 6*b*.

As one would expect from the composition, the yield point of alloy No. 3 is considerably higher at all temperatures than the other alloys. The yield point of No. 3, however, falls off more

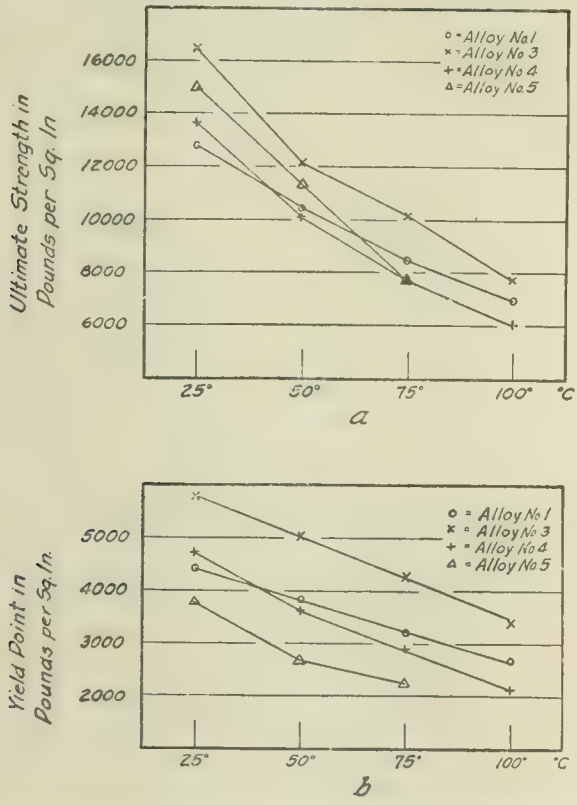

FIG. 6 *a* and *b*.—*Curves showing effect of temperature on yield points and ultimate strength of alloys Nos. 1, 3, 4, and 5*

rapidly than No. 1 with increasing temperatures. The points in both these cases appear to lie on a straight line. This is not the case with alloys Nos. 4 and 5, which contain lead. For both of these alloys the yield point seems to drop off more rapidly at first, between 25 and 50° C. It is significant to note that while the yield point of alloy No. 4 is higher than No. 1 at room temperature, it is lower at 50° C and decreases at a more rapid rate between 25 and 100° C than does No. 1, and that the yield point of tin-base alloys is higher at all temperatures above 50° C.

The yield point of alloy No. 2, curves of which are not given, is slightly higher at room temperature than that of No. 3, but at 50° C its yield point is slightly less than No. 3, and so, if the bearing heats to 50° C or over, any advantage gained by using No. 2 alloy in a bearing is lost in so far as the yield point is concerned.

There are given in Fig. 6*a* curves showing the variation of the ultimate strength with the temperature. Here, as with the yield point, alloy No. 3 has the maximum value throughout the temperature range, and alloys Nos. 1 and 3 maintain their strength better, having a higher ultimate strength at temperatures above 60° C than either alloys Nos. 4 or 5, which contain lead, even though the ultimate strength of No. 1 at room temperature is less than that of Nos. 4 or 5. The ultimate strengths of the four alloys at 100° C stand in the same relation to each other as their respective yield points.

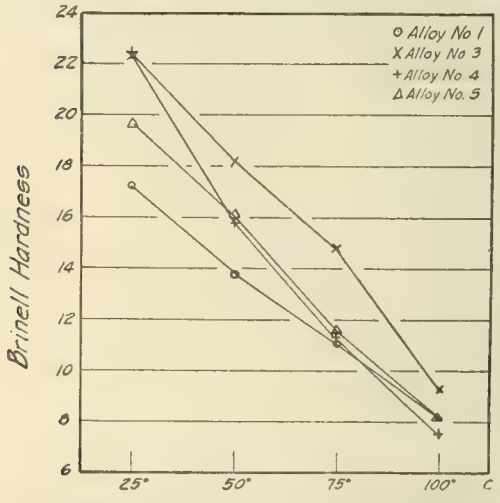

FIG. 7.— *Curves showing relation of Brinell hardness to temperature for alloys Nos. 1, 3, 4, and 5*

2. BRINELL HARDNESS TESTS

It is noted that the Brinell hardness values obtained for the tin-base alloys are considerably lower than those usually given. This difference may be due to the small percentage of impurities in the alloys used in this investigation as compared with similar alloys as ordinarily prepared.

Curves showing the variation of the Brinell hardness with temperature are given in Fig. 7. Here, again, alloy No. 3 has a maximum value throughout the temperature range. There is no evident relation between the relative magnitude of either the ultimate strength or yield point and the Brinell hardness.

The hardness of alloys Nos. 4 and 5, however, drops off very rapidly with increasing temperature, while Nos. 1 and 3 maintain their hardness, both having a greater hardness at 100° C than No. 4, and No. 1 having the same value as No. 5 at this temperature.

IX. EFFECT OF PROLONGED HEATING AT 100° C

Oftentimes a babbitted bearing which has given good service will for no apparent reason gradually become soft and "wipe out." As a working hypothesis it was thought that this failure with age might be due to softening from prolonged heating causing an annealing action. In order to determine the validity of this tentative hypothesis, compression specimens of alloys Nos. 1, 3, 4, and 5 were heated in an oil bath for from one to six weeks at temperatures between 90 and 100° C. They were then tested at room temperature with the results given in the following table:

TABLE 6.—Effect of Prolonged Heating on the Yield Point in Compression

Days heating at 100° C	Yield points of the various alloys			
	Alloy No. 1	Alloy No. 3	Alloy No. 4	Alloy No. 5
	Lbs./in.²	Lbs./in.²	Lbs./in.²	Lbs./in.²
0	4550	5750	4650	3750
7	4500	5500	.....	3450
14	4600	5800	4250	3200
28	5025	5650	4750	[a] 2800
42	4900	5950	4850	3150

[a] One specimen only. All other values are the average of two specimens.

A study of the above table indicates that for alloys Nos. 1, 3, and 4, heating at 100° C for 42 days has no appreciable effect on the value of the yield point when the specimens are cast in the manner indicated. For alloy No. 5, however, there is a very evident decrease in the value of its yield point with the prolonged heating which, however, evidently takes place in the first two weeks of the heating.

X. EFFECT OF SMALL PERCENTAGES OF LEAD

The specifications for high-grade, tin-base alloys such as Nos. 1, 2, and 3 call for a low lead content generally not to exceed 0.35 per cent.

Many believe, and one investigator [6] has presented experimental evidence, that percentages of lead even up to 5 per cent are not harmful but possibly beneficial. The authors have therefore investigated the effect of small percentages of lead on the yield point and ultimate strength of No. 2 alloy at room temperature and at 75° C.

[6] Jesse L. Jones; see footnote 1.

The alloys were prepared by adding metallic lead to the No. 2 babbitt in amounts shown by the chemical analysis given, together with the yield points, in Table 7.

TABLE 7.—Effect of Lead on Compressive Strength

Percentage of lead	Yield point at—		Ultimate strength at—	
	25° C	75° C	25° C	75° C
	Lbs./in.²	Lbs./in.²	Lbs./in.²	Lbs./in.²
0.00	6130	4000	15 175	9 395
0.26	5850	3700	15 640	10 010
0.51	5750	3300	14 025	9 765
1.01	6300	.....	.....	.....
1.25	6000	4100	16 380	10 600
5.04	5850	3850	15 330	9 725

The addition of amounts up to 5 per cent of lead to this babbitt seems to have no very appreciable effect on its mechanical properties in compression at room temperature or at 75° C under the conditions of test used. The authors think, however, that these tests should not lead to an increase in the lead content tolerance in tin-base bearing metal specifications until much more work is done along this line and particularly to determine the possible effect of small percentages of lead on the resistance to repeated impact.

XI. SUMMARY AND CONCLUSIONS

An apparatus is described for determining the yield point and ultimate strength of white metal bearing alloys at temperatures up to 100° C. A new design of heating apparatus is described for determining the Brinell hardness of metals at temperatures up to 100° C.

The results of compression tests and Brinell hardness tests at temperatures up to 100° C are given for five typical white metal bearing alloys, including three tin-base alloys, one lead-base alloy, and one intermediate alloy, which show that the tin-base alloys maintain their properties better at elevated temperatures than the lead-containing alloys.

Results of tests are given which indicate that the addition of amounts up to 5 per cent of lead in a high-grade babbitt does not affect the yield point or ultimate strength at 25 or 75° C.

The yield point of tin-base alloys is not affected by heating for six weeks at about 100° C, but the yield point is lowered in the lead-base alloy by heating for two weeks at about 100° C.

WASHINGTON, November 6, 1920.

DEPARTMENT OF COMMERCE

TECHNOLOGIC PAPERS

OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 189

METHOD FOR DIFFERENTIATING AND ESTIMATING UNBLEACHED SULPHITE AND SULPHATE PULPS IN PAPER

BY

R. E. LOFTON, Associate Physicist
M. F. MERRITT, Laboratory Assistant
Bureau of Standards

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METHODS FOR DIFFERENTIATING AND DETERMINING HYDROLYTIC TOLUENE AND TOLUENE FREE IN TAPIOCA

ISSUED JUNE 15, 1920

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1920

A METHOD FOR DIFFERENTIATING AND ESTIMATING UNBLEACHED SULPHITE AND SULPHATE PULPS IN PAPER

By R. E. Lofton and M. F. Merritt

ABSTRACT

The purpose of this paper is to fill a need felt especially by paper chemists and analysts for a rapid and reliable method of distinguishing between and of making an approximately correct quantitative determination of mixtures of unbleached sulphite and sulphate pulps. This paper gives briefly the basic differences in the manufacture of the two pulps, and contains a concise review of the methods that have been recommended from time to time for distinguishing between unbleached sulphite and sulphate pulps. It gives the procedure followed in developing a new and comparatively rapid method for distinguishing between these pulps, and also gives some of the more important experiments carried out with various stains during this investigation. The method of preparing the new stain and the method of procedure for differentiating between unbleached sulphite and sulphate fibers is described in detail, and tables showing the results of quantitative microscopical analysis of mixtures of these fibers stained by the new method are given.

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I. INTRODUCTION

The purpose of this publication is to give a review of the various methods proposed for distinguishing between unbleached sulphite and sulphate fibers, and especially to describe the development and application of a new method for accomplishing this result.

That there is and has existed for some time a demand for a quick and certain method for differentiating between these two pulps is indicated by the various attempts that have been made from time to time to develop such a method. To be satisfactory, any method proposed must be practical as well as certain—that is, it must not require any expensive apparatus, or special experience, or tedious and lengthy manipulation—but it must be such as may be carried out in a few minutes time by any one ordinarily familiar with the microscopical examination of paper-making fibers.

Such a method would be of service to pulp manufacturers and jobbers, to manufacturers of sulphite and sulphate papers, and to the retailers and consumers of sulphite and sulphate papers in aiding them to determine whether they are getting what they desire. It is useless for a jobber or a consumer to specifically order an all or part sulphate wrapping paper unless there is some means of determining whether the article ordered is being furnished.

The authors acknowledge the assistance of M. B. Shaw, who made a large number of fiber estimations, the results of which appear in Tables 3, 4, and 5, and who gave assistance as to methods of preparing the stains.

II. THEORETICAL CONSIDERATIONS INVOLVED

1. FUNDAMENTAL DIFFERENCES BETWEEN THE TWO PULPS

The difficulties to be overcome in developing a method for distinguishing between unbleached sulphite and sulphate pulps are due to the similarity of the pulps. Both are made, for the most part, from the same raw material, with but two or three exceptions. Spruce, hemlock, balsam fir, yellow pine, tamarack, and white fir are used in making both pulps. In addition to these woods, jack pine and cypress are often used in the sulphate process. Yellow pine is more generally used in the sulphate than in the sulphite process, because the former process is better adapted to woods rich in rosin and oil. The woods generally used in foreign countries are of the same kinds as those used in the United States and Canada except that black spruce is frequently used in Finland, Norway, and Sweden.

Since the raw material from which these pulps are made is in general the same, the only distinguishable differences between the pulps are to be found in the two different digesting processes employed in their manufacture.

Sulphite pulp is made by cooking wood chips in a solution of bisulphite of calcium, or of calcium and magnesium. This solution

has an acid reaction. The chips are cooked under a steam pressure of from 60 to 80 pounds for from 8 to 20 hours. The process was invented by B. C. Tilghman, of Philadelphia, who took out the first United States patent in 1867.

Sulphate pulp, frequently called "Kraft" (a German word meaning "strength"), is made by cooking wood chips in a solution the chief ingredient of which is sodium sulphide. The sodium "sulphate" added from time to time is reduced to "sulphide" during the preparation of the cooking liquor. The reaction of this solution is alkaline. In this process the chips are cooked under a steam pressure of about 100 pounds for from 2 to 7 hours. This process was invented by Carl F. Dahl, of Danzig, Germany, about 1883, and was introduced into America in 1907, when the Brompton Pulp and Paper Co. set up a sulphate mill in Canada.

Unbleached sulphite pulp is used in the manufacture of wrapping paper and bag stock, of many so-called Kraft and manila papers, of twines used in tying bundles and in making paper rugs, onion and potato sacks, and in any pulp or paper product where strength is the chief consideration. Sulphate pulp has a considerably darker color than sulphite, for which reason the latter is often colored in the process of being made into paper to resemble sulphate pulp. In general, sulphate pulp may be used wherever it is permissible to use unbleached sulphite. In cases where the greatest possible strength is required, sulphate is used instead of sulphite, since it is somewhat stronger. The cost of sulphate, due to a more expensive process of manufacture, is somewhat more than that of sulphite pulp.

2. POSSIBLE BASES OF DIFFERENTIATION

Commenting on the problem, C. W. Schwalbe stated[1] in 1914 that there was no simple method known at that time for distinguishing between unbleached sulphite and sulphate pulps; also, that it is difficult to develop such a method because the pulps are so closely related. Schwalbe recognized, too, as do others who have made a study of the two pulps, that there are two differences between them which may be used as a basis on which to develop reactions which will differentiate them, namely, (1) the difference in the amounts of incrusting or ligneous material, and (2) the different chemical changes which have been brought about by the different digesting processes.

[1] Testing Methods for Sulphite and Sulphate Cellulose in Paper, Pulp and Paper Magazine, p. 21; Jan. 1, 1914.

3. DIFFERENT AFFINITIES OF PULPS FOR DYES

It is pretty generally known by persons who have made any analytic study at all of paper-making pulps that those pulps differing in the degree of cooking, or of bleaching, or both, usually have quite different affinities for various dyes and stains. The dye most generally used to show the properties of different pulps in this respect is malachite green, a basic aniline or coal-tar dye which has a great affinity for highly lignified fibers and very little or no affinity for pulps and fibers freed from incrusting matter. The result of this property is that, when malachite green alone is used to stain a pulp composed of a mixture of fibers having a high content of lignin and of other fibers more thoroughly digested, one gets a range of shades varying from deep green to very light green, or, perhaps, to the entire absence of color in the case of fibers that are completely freed from incrusting matter. Although there may be no sharp color line of demarcation to set off any one group of fibers against any other, results of this kind may serve the purpose of determining the thoroughness or the uniformity of the cooking or of the bleaching action. But if this or a similar dye or stain is followed by one of a contrasting color, as recommended by Siebert and Minor,[2] it will sometimes be found possible to separate quite distinctly a mixture of two different pulps, as, for example, a well-cooked from a slightly-cooked pulp, or a well-bleached from a poorly-bleached pulp, or a chemical from a mechanical pulp. These authors, however, recommend that malachite green be followed by an acid dye, while in the method described below both of the dyes used are basic.

Whether the different affinities of the two pulps for various dyes and stains are due to different chemical properties of the pulps brought about by the different cooking methods, or whether they are due merely to the different amounts of incrusting matter remaining in the sulphite and sulphate pulps, is a question on which cellulose chemists do not agree.

III. METHODS PROPOSED BY EARLIER EXPERIMENTERS

In studying this problem the available methods proposed from time to time by others who have experimented along this line were tried. Below is given a description of each method, together with a brief statement of results obtained by the writers in trying the method.

[2] The Differentiation of Sulphite Pulps, Paper, 25, No. 21; Jan. 28, 1920.

1. KLEMM'S METHODS

Klemm is given credit for developing two methods, in one of which use is made of malachite green alone, and in the other use is made of both malachite green and rosaniline sulphate.

Klemm's malachite green method is described by R. W. Fannon [3] as follows:

> The reagent consists of a saturated solution of malachite green, a coal-tar dye, to which 2 per cent acetic acid has been added. A portion of pulp is treated with a few drops of the reagent sufficient to soak it, and the excess blotted up. The fiber is then examined microscopically. Unbleached sulphite is colored full green; sulphate is colored light green.

The results obtained by using this method were not satisfactory, since there is no sharp or distinct line of demarcation between the two pulps, some of the more deeply colored sulphate fibers showing a deeper green than some of the lighter colored sulphite fibers. These results are in harmony with the statements made above regarding the use of malachite green in staining various pulps and fibers.

Schwalbe mentions another procedure,[4] a malachite green and rosaniline-sulphate method, which he also calls Klemm's method. No details as to making up the dye solutions or method of applying them are given.

It was found, however, that a fairly good differentiation between the two pulps is had if they are stained for about two minutes with a one-half per cent aqueous solution of malachite green, rinsed with water, then stained with a solution of rosaniline sulphate acidified with sulphuric acid. The same results may also be obtained if these two stains are compounded in the right proportions before being applied to the pulps.

When the chemical composition of rosaniline sulphate is compared with that of magenta, or fuchsine, one of the dyes recommended below, it is evident that they are very closely related, and that the color radical is the same in both. An insight into the constitution of these compounds may be had from any good treatise on organic chemistry.

Pararosaniline and rosaniline [5] are the bases of the fuchsine dyes. Both are triacid bases, stronger than ammonia. The basic fuchsine or magenta dyes are formed by treating these bases with various acids, particularly hydrochloric and acetic,

[3] An effort to find a simple means of differentiation between sulphite and sulphate pulp. (A thesis prepared at the University of Maine, June, 1916.)

[4] Pulp and Paper Magazine, p. 21; Jan. 1, 1914.

[5] A. Bernthsen, Organic Chemistry, translation by J. J. Sudborough; New York, 1912.

water being eliminated during the reaction. The formula of rosaniline is

$$C(OH)\left\langle\begin{matrix}C_6H_4.NH_2\\C_6H_4.NH_2\\C_6H_3(CH_3).NH_2\end{matrix}\right\rangle \text{ or } C_{20}H_{21}N_3O.$$

By treating with hydrochloric acid, the following reaction takes place:

$$\underset{\text{Rosaniline}}{C_{20}H_{21}N_3O} + \underset{\substack{\text{Hydro-}\\\text{chloric}\\\text{acid}}}{HCl} = \underset{\substack{\text{Rosaniline}\\\text{hydrochloride}}}{C_{20}H_{20}N_3Cl} + \underset{\text{Water}}{H_2O}.$$

From analogy, it appears that a similar reaction takes place when rosaniline is treated with sulphuric acid, and that rosaniline sulphate is essentially a basic fuchsine or magenta dye, but with an acid radical not usually found in dyes put out under this name.

$$\underset{\text{Rosaniline}}{(C_{20}H_{21}N_3O)_2} + \underset{\substack{\text{Sulphuric}\\\text{acid}}}{H_2SO_4} = \underset{\substack{\text{Rosaniline sul-}\\\text{phate}}}{(C_{20}H_{20}N_3)_2SO_4} + \underset{\text{Water}}{2H_2O}.$$

As the coloring power, however, of these dyes lies wholly in the basic constituent, there can be no doubt but that rosaniline sulphate is essentially basic fuchsine or magenta.

One authority [6] states that basic magenta is a mixture of the hydrochloride and acetate of rosaniline and pararosaniline; according to another, it is a mixture of the hydrochlorides of rosaniline and pararosaniline. The formula of pararosaniline is

$$C(OH)\left\langle\begin{matrix}C_6H_4.NH_2\\C_6H_4.NH_2\\C_6H_4.NH_2\end{matrix}\right\rangle \text{ or } C_{19}H_{19}N_3O.$$

The reaction of this base with acids is like that of rosaniline.

The malachite-green and rosaniline-sulphate stain, however, does not give as brilliant color contrast between the two kinds of fibers. This advantage of the use of basic fuchsine over that of rosaniline sulphate is probably explained by the greater solubility of basic magenta as compared with that of rosaniline sulphate, since it was found by test that basic magenta is much more soluble in water than is rosaniline sulphate. This fact is in harmony with the known properties of the salts of these acids.

[6] Thomas H. Norton artifical dyestuffs used in the United States, Special Agent Series, No. 121, Bureau Foreign and Domestic Commerce, Washington D. C. Ira Remsen, Organic Chemistry, Fifth revision; New York, 1909.

2. SCHWALBE'S METHODS

Schwalbe also describes [7] two methods of analysis, in one of which aqueous solutions of ferric chloride and potassium ferrocyanide are used, and in the other solutions of a copper salt and an organic dye. The former method is also briefly outlined by Fannon.[8]

The ferric-chloride and potassium-ferrocyanide method consists in extracting the pulps for about 30 minutes in alcohol and ether to remove rosin, treating with a 0.05 *N* solution of ferric chloride at from 60 to 80° C for about 30 minutes, or until all the fibers settle to the bottom of the container, washing with distilled water, and treating with a 1 per cent solution of sulphuric acid, to which is then added from four to eight drops of a 2 per cent solution of potassium ferrocyanide. The container and contents are then placed on a water bath and kept at a temperature of from 60 to 80° C for from 5 to 10 minutes. The pulps are then washed and examined under the microscope. Sulphite fibers are colored a deep blue, and sulphate fibers a much lighter blue.

This method did not give results that would enable a distinction to be made in all cases between the two pulps, nor to permit estimations in any case. Moreover, the process is too long and tedious to be practical. However, since Schwalbe so highly recommends this method, it is thought probable that better results could be obtained if enough time should be spent in studying and experimenting with it.

By the copper method the pulps are boiled in a solution of a copper salt such as copper sulphate, washed, then treated with a solution of an organic dye such as benzopurpurine 10B. The benzopurpurine treatment develops in some classes of fibers an intensely blue color, according to Schwalbe, due to the formation of a so-called copper lake with the copper salt retained by the fibers.

This method was not experimented with, since Schwalbe himself condemns it as being too involved and uncertain in results, especially with pulps so similar in their properties as unbleached sulphite and sulphate.

---

[7] Pulp and Paper Magazine, p. 21; Jan. 1, 1914.

[8] Thesis referred to above.

3. FANNON'S METHODS

Fannon [9] found that a saturated aqueous solution of rosaniline sulphate, to which is added 1 to 3 per cent of alcohol and enough sulhpuric acid to cause the solution to take on a violet shade, is a satisfactory stain. He used two methods of applying this stain and examining its staining action:

In the first method, two or three drops of the solution were allowed to fall in the same spot onto the samples of sulphite and sulphate pulps tested, and the color reactions observed with the unaided eye. The unbleached sulphite pulps showed "a deep bluish center with a yellow ring surrounding," while the sulphate sample gave "a deep red coloration." "From 7 to 15 minutes after the stain was dropped on, the distinctive colorings were evident. After an hour or so the colors faded."

In the second method, "mixtures of sulphate fibers and sulphite fibers were made, stained with the last-named solution. The different fibers developed their respective colorations, and under the microscope estimations of each were possible."

This stain was not found satisfactory when used according to either method recommended, because the color differences shown by the two pulps are not sharp enough to enable one to detect without doubt even the presence of small percentages of one pulp when mixed with the other.

IV. THE MALACHITE-GREEN AND FUCHSINE METHOD

1. SOURCES OF MATERIALS USED

The samples of pulps and papers used in this investigation consisted of 188 pieces collected from a number of different firms, and represent pulps manufactured in all parts of this country, and in at least four foreign countries, Canada, Finland, Norway, and Sweden. All possible information was obtained regarding these samples, such as the kind of wood used and in what locality grown, the degree of cooking, etc.

The dyes and stains used during the earlier part of this investigation were obtained chiefly from the chemical division of this Bureau. Most of them had been in stock for some time, and all, or nearly all, were of German manufacture. But after the method given below had been pretty thoroughly worked out, samples of the two dyes used in this method were obtained from different sources in this country in order to determine whether the different makes of dyes were equally suitable for the purpose.

[9] Thesis referred to above.

2. METHOD OF ATTACK AND EARLIER EXPERIMENTS

The method of conducting the investigation was to try out on the two kinds of pulps the reaction of various biological stains and dyes recommended by authorities on microscopical methods, and also other stains and dyes that were suggested in one way or another. Most of the experimenting was carried on with red, green, and blue dyes, since dyes of these colors are usually the most brilliant and positive in their coloring action. The few yellow dyes that were tried did not give sufficiently brilliant color effects on the fibers, and little work was done with them.

In making this investigation by what may be properly called the empirical method, a great number of tests were necessarily made that are relatively valueless and therefore are omitted from this publication. Those which may be of value to others who are experimenting along the same line are given in Table 1.

TABLE 1.—Interesting Experiments Made and Stains Used with the Color Reactions on Sulphate and Sulphite Fibers

[Ext. alc.=extracted with alcohol; aq.=aqueous; so'.=solution.]

Serial No.	How stained	Action on—		Remarks
		Sulphate	Sulphite	
2	Safranin in equal parts water and alcohol, rinsed with water, Delafield's hematoxylin.	Colorless to faint reddish brown.	Pink to light magenta.	Color difference quite noticeable.
6	Ext. alc., boiled 1 minute, ¼ per cent aq. sol. malachite green, rinsed with water, rosaniline sulphate and little sulphuric acid.	Clear blue, bundles green.	Pale gray to lavender.	Differentiation good.
7	Ext. alc., boiled 1 minute, ¼ per cent aq. sol. malachite green, rinsed with water, eosin in equal parts alcohol and water.	Dark blue.....	Colorless to faint purple.	Pits of sulphite fibers colored green.
13	Ext. alc., heated to 100°C in ¼ per cent aq. sol. malachite green, rinsed with water, 1 per cent aq. alc. sol. safranin.	Light to dark red.	Light to dark red.	Do.
17	Boiled 1 minute in 1 part rosaniline sulphate sol., then 1 part 1 per cent aq. sol. malachite green added, lightly rinsed with water.	Blue.....	Purple.....	Good differentiation.
44	Ext. alc., boiled in water, acidified solution 50 per cent alcohol, Delafield's hematoxylin 10 to 15 minutes, rinsed with water, ½ per cent sol. congo red 1 minute, rinsed with water.	Red.....	.....do.....	Fair differentiation.
45	Ext. alc., boiled in water, acidified solution 50 per cent alcohol, Delafield's hematoxylin left on until air dry, rinsed with water, ½ per cent sol. congo red 1 minute, rinsed with water.	.....do.....	Purple with little red.	Do.

TABLE 1.—Interesting Experiments Made and Stains Used with the Color Reactions on Sulphate and Sulphite Fibers—Continued

Serial No.	How stained	Action on—		Remarks
		Sulphate	Sulphite	
57	Ext. alc., washed with 1 per cent aq. sol. tannic acid, then equal parts 1 per cent aq. solutions acid fuchsine and malachite green 3 minutes, rinsed with water.	Blue.....	Purple.....	Good differentiation.
79	Ext. alc., boiled in water, dried with filter paper on slide, then 50 parts each of 1 per cent aq. solutions magenta and malachite green and 1 part 1 per cent aq. sol. tannic acid 2 minutes, quickly rinsed in 50–50 aq. alc. sol. slightly acidified with HCl, rinsed with water.	Greenish blue..	.....do.....	Differentiation excellent.
82	Boiled in water, ½ per cent aq. sol. malachite green and aq. sol. rosaniline sulphate and little H_2SO_4, all left on fibers 2 minutes, then rinsed with water.	Blue.	.....do.....	Color difference not as decisive as when magenta is used in place of rosaniline sulphate.
84	Slide as made up by No. 79 examined under microscope in polarized light, with both parallel and crossed nicols.	Various colors...	Various colors.	No distinguishing properties appeared.
85	Boiled in water, pulped and examined under microscope in polarized light with both parallel and crossed nicols.	Yellow, orange, and blue.	Yellow, orange, and blue.	Do.

Some of the pulps were extracted with alcohol to remove resins, and this fact is indicated in each case in the table. The pulps were then rinsed and cooked, at first, in clear water, and pulped, no chemical being used in the cooking process, as it was thought that any chemical treatment might possibly interfere with the staining action to follow. Experiments that were conducted later, however, indicated that cooking the sample of paper in a one-half per cent aqueous solution of caustic soda does not have any effect on the action of the dyes. The alcohol used at various times and for various purposes, as indicated, was in all cases ethyl alcohol. The color appearances and other characteristics noted are those which appeared under the microscope.

3. DETAILED METHOD OF USING

The stain which was found to be most satisfactory in differentiating between unbleached sulphite and sulphate pulps or fibers was a mixture of one part of a 2 per cent aqueous solution of malachite green and two parts of a 1 per cent aqueous solution of

basic fuchsine, or magenta. The solutions were made up according to the following formulas, kept in tightly stoppered separate bottles, and mixed only when wanted for use:

A—Malachite green	2 g
Distilled water	100 cm^3
B—Basic fuchsine	1 g
Distilled water	100 cm^3

Since there is considerable variation in the quality of dyes from various sources, it is not to be expected that any given combination of dyes or method of procedure will best fit all cases; it is, indeed, more than probable that the compound stain will have to be modified somewhat as to its two components, depending on the source of the dyes.

After this stain, therefore, has been made up according to formula, it will be necessary to test it out on samples of sulphite and sulphate fibers. To do this, samples of unbleached sulphite and sulphate pulps should be prepared and a few fibers of each placed on a slide, care being taken not to get the two samples mixed. The fibers are then dried and stained, as directed below, and then examined under the microscope. All the sulphate fibers should have a blue or blue-green color, and all the sulphite fibers should have a purple or lavender color. If any purple fibers appear in the sulphate pulp this indicates that too much fuchsine is present in the combination, and a little more malachite green solution must be added to counteract this effect. If, on the other hand, some of the sulphite fibers show green or blue, there is too much malachite green in the combination, and more fuchsine solution must be added. Of course the analyst must be sure that he is using authentic samples of the two pulps for this test. When tested out in this manner and the proper combination found, the stain is ready to be used on unknown combinations of fibers containing either unbleached sulphite or sulphate, or both.

A mixture of one-half sulphite and one-half sulphate may also be used to test out the stain, the proper combination for the stain being indicated when one-half of the fibers are colored blue, and the other half purple.

The stain should not be used for more than a few hours after being compounded and should be made up anew at least each day.

The method of preparing the samples of pulps or papers for staining, and of applying the stain, is as follows:

The sample is boiled for a few minutes in water or in a one-half per cent aqueous solution of sodium hydroxide, and the fibers are thoroughly disintegrated by shaking in a test tube or other receptacle about half filled with water, glass beads being added if the fibers can not otherwise be separated. Several fibers are then removed by means of a teasing needle, or preferably by means of a glass tube [10] about seven thirty-seconds of an inch in diameter, placed on a microscope slide, and dried by the use of hard filter or blotting paper. Two or three drops of the compound stain are then placed on the fibers by means of a suitable dropper or a pipette and allowed to remain 2 minutes, during which time the fibers are being teased apart and moved about in the stain on the slide. This teasing is necessary in order that the stain may have equal opportunity to act on all the fibers. At the end of 2 minutes the excess stain is removed with three or four thicknesses of hard filter paper, and the fibers treated with three or four drops of a weak aqueous solution of hydrochloric acid, made by adding 1 cm^3 of concentrated acid (sp. gr. 1.19; HCl 37 per cent) to 1 liter of distilled water. The acid solution is allowed to remain on the slide for from 10 to 30 seconds, during which time the fibers are teased and moved about rapidly. Following this, the excess acid solution is removed with filter paper, three or four drops of distilled water applied, the fibers quickly teased about, and the water absorbed with filter paper. If all the excess stain has been removed from the slide at this point, a drop or two of water may be added, the fibers spread about on the slide, and a cover glass placed over them. But if too much stain remains on the slide at this point, it will be necessary to rinse again with distilled water before applying the cover glass. After the cover glass has been placed in position the fibers are ready for examination under the microscope.

4. ESTIMATING PERCENTAGES

The color contrast not only enables one to detect the presence of one or both of these fibers, but is sharp enough to enable one, after some practice, to make an approximately correct estimate of the percentages of each of these fibers present.

[10] F. C. Clark, Paper Testing Methods, Tappi Publishing Corp., New York, N. Y.; 1920.

In order to get a practical idea as to what could be accomplished in estimating the percentages of sulphite and sulphate fibers when stained as directed, three persons, all of whom had had experience in estimating the percentages of fibers stained with the zinc-chloride and iodine stain, made a number of estimations on known mixtures of these two pulps.

Seven mixtures, made from representative samples of sulphite and sulphate pulps, were prepared by weighing on a chemical balance the proper proportions of each pulp, the weights of the two components totaling 30 g in each case. The pulps were then thoroughly mixed by agitating in a tight container with considerable water. The series made up contained 20, 25, 40, 50, 60, 75, and 80 per cent of unbleached sulphite. A representative portion of each member of this series was placed in a container and labeled. Microscope slides were then made up from each of the seven samples, each slide being given an unknown mark of identification, and handed over to the analysts for their estimations. Four different series of estimates were made, in each of which a different make of American dyes was used.

V. SUMMARY OF RESULTS

In the tables below are given the results of estimates on the seven fiber mixtures, each table showing the results obtained by using dyes from one of the four sources.

TABLE 2.—Results of Analyses, Using Dyes from First Source

Number of estimations	Observer	Percentage sulphite in mixture				Error of average	Probable error (r)	Huge error (u)
		Estimated			Actual			
		Maximum	Minimum	Average				
						Per cent	Per cent	Per cent
16	A	40	20	23.8	20	+3.8	5.1	12.3
14	C	50	10	27.9	20	+7.9	8.8	21.6
16	A	50	20	33.1	25	+8.1	8.2	20.0
14	C	50	20	33.9	25	+8.9	8.7	21.2
16	A	60	25	41.6	40	+1.6	7.1	17.3
14	C	90	25	50.7	40	+10.7	12.7	30.9
16	A	75	40	51.6	50	+1.6	5.4	13.2
14	C	80	40	57.1	50	+7.1	9.5	23.1
16	A	80	60	67.2	60	+7.2	7.1	17.3
14	C	90	60	69.6	60	+9.6	8.7	21.2
16	A	90	25	73.7	75	−1.3	9.6	23.4
14	C	80	40	72.9	75	−2.1	7.8	18.9
16	A	95	75	80.6	80	+0.6	3.4	8.2
14	C	90	25	76.4	80	−3.6	10.1	24.7

TABLE 3.—Results of Analyses, Using Dyes from Second Source

Number of estimations	Observer	Percentage sulphite in mixture				Error of average	Probable error (r)	Huge error (u)
		Estimated			Actual			
		Maximum	Minimum	Average				
						Per cent	Per cent	Per cent
16	A	40	20	24.7	20	+4.7	6.0	14.7
2	B	25	25	25.0	20	+5.0	3.4	8.2
5	C	40	20	24.0	20	+4.0	6.0	14.7
16	A	50	20	37.2	25	+12.2	10.0	24.4
3	B	40	25	35.0	25	+10.0	8.3	20.2
5	C	25	20	24.0	25	−1.0	1.5	3.7
16	A	75	25	50.0	40	+10.0	10.5	25.5
3	B	60	50	53.3	40	+13.3	9.5	23.3
6	C	75	30	47.5	40	+7.5	10.8	26.3
16	A	75	50	54.1	50	+4.1	5.4	13.2
3	B	60	60	60.0	50	+10.0	6.7	16.5
6	C	60	40	51.7	50	+1.7	6.2	15.0
16	A	80	50	65.0	60	+5.0	7.6	18.6
3	B	75	50	66.7	60	+6.7	9.1	22.3
5	C	60	50	56.0	60	−4.0	4.3	10.4
16	A	80	60	75.6	75	+0.6	4.3	10.5
3	B	80	80	80.0	75	+5.0	3.4	8.2
5	C	80	70	77.0	75	+2.0	3.0	7.4
16	A	80	75	78.4	80	−1.6	1.9	4.6
3	B	80	75	78.3	80	−1.7	2.0	4.8
4	C	80	70	73.7	80	−6.3	5.1	12.4

TABLE 4.—Results of Analyses, Using Dyes from Third Source

Number of estimations	Observer	Percentage sulphite in mixture				Error of average	Probable error (r)	Huge error (u)
		Estimated			Actual			
		Maximum	Minimum	Average				
						Per cent	Per cent	Per cent
16	A	50	20	27.5	20	+ 7.5	8.8	21.4
16	B	40	20	26.9	20	+ 6.9	7.1	17.2
16	A	75	20	32.5	25	+ 7.5	10.6	25.8
16	B	50	20	37.2	25	+12.2	11.0	27.0
16	A	60	25	43.1	40	+ 3.1	8.0	19.5
16	B	75	25	48.4	40	+ 8.4	9.9	24.3
16	A	60	20	45.3	50	− 4.7	8.1	19.9
16	B	75	40	55.3	50	+ 5.3	7.6	18.5
16	A	75	25	55.0	60	− 5.0	8.0	19.5
16	B	80	40	63.8	60	+ 3.8	8.1	19.8
16	A	80	60	74.4	75	− 0.6	4.9	12.0
16	B	80	50	70.9	75	− 4.1	6.3	15.3
16	A	80	50	75.3	80	− 4.7	6.4	15.5
16	B	80	70	78.4	80	− 1.6	2.2	5.4

TABLE 5.—Results of Analyses, Using Dyes from Fourth Source

Number of estimations	Observer	Percentage sulphite in mixture				Error of average	Probable error (r)	Huge error (u)
		Estimated			Actual			
		Maximum	Minimum	Average				
						Per cent	Per cent	Per cent
17	A	25	20	20.6	20	+0.6	1.2	2.8
17	B	25	20	20.6	20	+0.6	1.2	2.8
17	A	40	20	28.2	25	+3.2	5.0	12.1
17	B	50	20	27.1	25	+2.1	4.8	11.8
17	A	60	25	40.6	40	+0.6	5.3	12.9
17	B	50	25	39.7	40	−0.3	3.0	7.2
17	A	60	40	48.8	50	−1.2	3.3	8.0
17	B	75	40	50.9	50	+0.9	5.5	13.4
17	A	75	25	58.8	60	−1.2	7.7	18.7
17	B	75	50	59.4	60	−0.6	4.8	11.6
17	A	80	60	74.1	75	−0.9	4.7	11.5
17	B	80	60	74.1	75	−0.9	4.7	11.5
17	A	80	75	78.2	80	−1.8	2.0	4.9
17	B	80	75	78.5	80	−1.5	1.8	4.5

The analysts, or persons who made the estimations, are referred to as "A," "B," and "C."

The error of averages is the difference between the average of all estimations for that particular case and the actual percentage of sulphite in the mixture. The average result is computed from a number of estimations in most cases; the exact number of estimations in each case is given in the tables.

The most probable error is an error of such magnitude that a single estimation has an equal chance of being either within the error or outside of it. The given errors are computed by the method of least squares according to the formula

$$r = 0.6745\sqrt{\frac{\Sigma X^2}{n}},$$

in which r is the most probable error, X is the error of one estimation, ΣX^2 is the sum of the squares of all errors of n estimations, and n is the total number of estimations made on that particular mixture by the given analyst.

The huge error is an error of such magnitude that the chances are equal that 9 estimations out of 10 will be within the error, and only 1 in 10 greater than it. These results are computed by the method of least squares, according to the formula

$$u = 1.65\sqrt{\frac{\Sigma X^2}{n}}, \text{ or } u = 2.44r,$$

in which u is the huge error.

In no case did an individual analyst make more than one estimation on the same miscroscope slide or the same field of fibers.

It is noteworthy that the errors in the last series, or table, are much less than those in the first three series. This is no doubt due in large part to the differences between the dyes used in the first three series and those used in Table 5, since those used in the last series gave much more satisfactory color effects than those used in the first three series. This result is also certainly due in part to the greater familiarity and experience the analysts had with the stain at the time the last series was run.

There are at least two criticisms that may be offered against these estimations. The first is that the results given may not quite represent estimates on wholly unknown mixtures, since each analyst knew in what proportion each of the seven combinations in the series was made up; although, as stated above, each sample was marked in code so that the analyst could not know what combination was being examined. When a study of the individual estimates is made, it appears that this circumstance did have an appreciable influence on the estimates on the mixtures near either end of the series. But since 0 per cent and 100 per cent are limits in all mixtures, it seems probable that estimates on mixtures near these limits will always be more nearly correct than estimates on mixtures containing a higher percentage of the smaller constituent.

The second criticism is that it is not practical in most cases to make 16 or 18 different estimates on the same sample of pulp or paper. This criticism is valid, especially as applied to mill conditions; for the paper-mill chemist is usually too busy to give so much time to the study of one sample of pulp or paper. It is necessary, however, that anyone, whether novice or expert, doing work of this kind, shall become familiar by actual experience with the use and properties of any stain before reliable results in estimating percentages can be had. To get best results in estimating, it is also necessary that one keep constantly in practice, and refer often to standard or known mixtures, accurately made up, and kept always at hand. When the analyst becomes familiar with the characteristics and use of the stain described above, it is probable that estimates can be made by its use as quickly and with as much accuracy as with the zinc-chloride and iodine stain.

WASHINGTON, November 27, 1920.

DEPARTMENT OF COMMERCE

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OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 190

"BLACK NICKEL" PLATING SOLUTIONS

BY

GEORGE B. HOGABOOM, Electroplating Inspector
T. F. SLATTERY, Electroplater
L. B. HAM

Bureau of Standards

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"BLACK NICKEL" PLATING SOLUTIONS

By George B. Hogaboom, T. F. Slattery, and L. B. Ham

ABSTRACT

To produce the so-called "Government bronze" finish on military hardware, "black nickel" plating was frequently applied. Investigation showed that for this purpose very complicated solutions were frequently employed, and at times great difficulty was encountered in producing uniform results. This paper describes the results of a few experiments on such solutions and contains recommendations regarding the composition and conditions of operation which will yield satisfactory deposits.

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1. INTRODUCTION[1]

During the war a very considerable demand arose for a black finish, the so-called "Government bronze," on military hardware and equipment. Owing to the difficulty of specifying definitely the color or other desired properties of the deposit, or the composition of solutions or methods of deposition, there was frequent uncertainty and even confusion in the various specifications and methods of inspection. In consequence numerous inquiries for information and assistance were received by the Bureau of Standards, directly and through military channels. The present paper is simply a summary of experiments and observations made and conclusions reached in the effort to answer such inquiries.

2. GENERAL PRINCIPLES

The principal formulas proposed for "black nickel" solutions[2] fall roughly into two classes—the alkaline cyanide baths and the neutral or slightly acid baths. Arsenic is usually an essential

[1] These notes were first published in the Monthly Review of the American Electroplaters' Society, July, 1919.

[2] For a summary of such formulas, see O. P. Watts, Metal Industry, 12, p. 24; 1914.

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constituent of the alkaline baths; in fact, as has been shown and is confirmed by this work, the deposits from such baths may consist principally of arsenic, thus approaching in composition and properties the deposits obtained from "arsenic black" baths (prepared by the solution of arsenic trioxide in sodium hydroxide and sodium cyanide). Such deposits, therefore, hardly justify the term "black nickel," except as an indication that the solutions contain (or originally contained) nickel salts and are operated with nickel anodes. The neutral baths contain nickel sulphate or nickel ammonium sulphate, together with salts of zinc, copper, etc., and usually some sulphur compound, such as a sulphite, thiosulphate (hypo), or sulphocyanate. Of these, the sulphocyanate solutions have been most frequently used. The experiments and observations of the Bureau were confined chiefly to the alkaline cyanide baths and the sulphocyanate baths, which will be considered separately.

3. ALKALINE CYANIDE SOLUTIONS

It is probable that these baths were originally prepared (as they still are in some cases) by adding to a portion of the regular nickel-plating bath a solution of sodium arsenite—that is, arsenic trioxide dissolved in sodium hydroxide—forming a precipitate (essentially nickel arsenite) which was then dissolved by the addition of sufficient sodium cyanide. It is, therefore, not surprising that difficulty was experienced when efforts were made to state a definite formula for preparing such solutions from the original chemicals. In order to obtain clear solutions, or to produce the particular shade of deposit desired, other additions were made to the solutions, which resulted in very complicated and even irrational formulas. Perhaps the best illustration of such a formula is the so-called Rock Island formula (which, however, probably did not originate at the Rock Island Arsenal).

SOLUTION NO. 1

1. Dissolve 6 pounds of double nickel salts in 6 gallons of water.
2. Add 4 ounces of sulphuric acid.
3. Add 1 pound of "phosphoric zinc" (prepared by dissolving zinc in sirupy phosphoric acid diluted with an equal volume of water).
4. Add 5 pounds of carbonate of ammonia (a small amount at a time).
5. Add 4 pounds of sodium cyanide.

SOLUTION NO. 2

1. Dissolve 10 pounds of arsenious acid and 10 pounds of caustic soda in 5 gallons of warm water.

Mix solutions No. 1 and No. 2 and dilute to 12 gallons.

Whatever the origin of this or equally complicated solutions suggested for military supplies, it is obvious that their use could not lead to satisfactory results. One of the difficulties experienced in attempting to follow some of the suggested formulas was that most of the metal in the salts was precipitated to the bottom, forming a sludge of cyanides, carbonates, or arsenites of nickel, zinc, etc. In one case after thorough mixing of the ingredients, only 7 per cent of the nickel added was found in the solution. Such solutions are, therefore, very wasteful, as well as unsatisfactory in operation.

In the effort to obtain a simple, satisfactory, alkaline "black nickel" solution, a large number of experiments were conducted, with frequent analyses of the solutions and the deposits. Assuming that the nickel is present as the double cyanide, and the arsenic as sodium arsenite, the simplest solution would consist of nickel sulphate treated with sufficient cyanide to dissolve the precipitate first formed, and arsenic trioxide with sufficient sodium hydroxide to dissolve it. Such solutions yielded, however, only unsatisfactory grayish deposits, which consisted principally of arsenic. When zinc sulphate was present in the solution more satisfactory deposits were obtained.

The exact function of the zinc is difficult to determine, since very little enters into the deposit. In the effort to employ certain of the commercial formulas, it was found that a very considerable excess of cyanide was necessary to obtain a clear solution. From such solutions, containing, for example, the ingredients here listed, good gray-black deposits were obtained.

Ingredients	Ounces per gallon	Grams per liter
Nickel ammonium sulphate (double salt)	6	45
Zinc sulphate	.5	3.8
Ammonium carbonate	5	37.5
Arsenic trioxide	.7	5
Sodium hydroxide	.7	5
Sodium cyanide	6	45

On continued operation, however, the deposits became lighter in color, a large amount of sludge formed in the solution, and the latter changed very materially in composition, losing both arsenic and zinc. This is to be expected, since in these solutions nickel anodes are used and there is no opportunity for the deposited arsenic and zinc to be replaced. The nickel content decreases less rapidly, although nickel anodes are not readily corroded in cyanide solutions.

The net result of a large number of experiments and analyses upon alkaline "black nickel" baths was the conclusion that no bath of this type can produce uniformly satisfactory results unless it is controlled by very frequent and decidedly difficult chemical analyses, such as are not usually available in commercial plating plants. We do not feel justified, therefore, in recommending any formula for such solutions.

4. SULPHOCYANATE SOLUTIONS

A typical formula of this type, which has been used for many years and is given in numerous texts on plating, is as follows:

Ingredients	Ounces per gallon	Grams per liter
Nickel ammonium sulphate (double salt)	8	60
Zinc sulphate	1	7.5
Sodium [a] sulphocyanate	2	15

[a] Originally potassium or ammonium sulphocyanate was used commercially, but recently the sodium salt has almost entirely replaced it with equal satisfaction. (See C. H. Proctor, Metal Industry, 16, p. 26; 1918.)

Since it has been found in practice that such solutions give entirely satisfactory black deposits, the experiments conducted were directed to securing simple means of control of these solutions rather than to securing new or possibly better formulas.

Analyses of deposits produced from such solutions when freshly prepared showed a considerable variation in composition, due no doubt to slight differences in the original neutrality of the solutions and to slight fluctuations in the voltage.

In this, as well as in the alkaline "black-nickel" solutions, even slight changes in voltage produce marked differences in the proportion of the metals precipitated and hence in the color and physical properties of the deposits. Thus in the above sulphocyanate solution a good black deposit is produced with 0.5 to 0.7 volt (between anode and cathode); while if the voltage be in-

creased to 1.5 to 2 volts, a light-colored, bright nickel deposit is secured. In the operation of such solutions, therefore, control of the voltage is more important than is the control of the current density. The latter probably will not, under good conditions, exceed 1 ampere per square foot (0.11 amp./dm^2).

Good "black-nickel" deposits were found to contain from 40 to 60 per cent of nickel, 20 to 30 per cent of zinc, 10 to 14 per cent of sulphur, and 10 per cent or more of organic matter of undetermined composition. (The organic matter was formed by the the decomposition of the sulphocyanate upon electrolysis.) The ratio of sulphur to nickel was always less than that required for nickel sulphide, which contains about 65 per cent of nickel and 35 per cent of sulphur. It therefore appears reasonably certain that the black deposit consists essentially of a mixture of nickel sulphide, nickel, zinc, and organic matter.

If the zinc content of the solution is increased very much above that indicated by the above formula, grayish deposits, with high zinc content, are produced. If, on the other hand, the zinc content of the solution is greatly reduced or actually eliminated, the deposit is nearly pure nickel, with a light or bright nickel color. It is, therefore, important to maintain a fairly constant composition of solution, although good results can be obtained over a moderate range. In the normal operation of such a solution, the changes that are likely to occur are (*a*) a decrease in sulphocyanate, which must, therefore, be replenished at intervals; (*b*) a decrease in zinc content; and (*c*) an increase in nickel content. The latter arises from the fact that with nickel anodes, nickel is going into solution more rapidly than it is deposited, since part of the current is used in depositing zinc. At first it might appear possible to replenish the zinc and prevent an increase in nickel content by the substitution of zinc anodes for part of the nickel anodes. Attempts to do this, however, resulted in too high a zinc content, owing to the fact that zinc anodes are so much more readily corroded than nickel. The most practical way, therefore, to reduce the nickel content is to use nickel anodes which do not corrode very readily—for example, those with 96 per cent or more of nickel—and, if necessary, to restrict the anode surface.

In general, and especially when using high percentage nickel anodes, there is a tendency for the solution to become acid. Among the various additions which have been proposed to maintain neutrality are ammonia and nickel carbonate. Neither is

entirely satisfactory, since addition of ammonia tends to precipitate the zinc from the solution unless great care is used, and, on the other hand, the use of nickel carbonate tends to still further increase the nickel content. For this purpose zinc carbonate should, therefore, be more satisfactory, since to the extent that it neutralizes the solution the zinc content will be maintained. Experiments have shown that by having an excess of zinc carbonate always present, good deposits may be secured over a considerable period. Whether in any case its use is sufficient to maintain the zinc content will depend upon the conditions of operation, and can be determined only by occasional analyses. Nickel, zinc, ammonia, and sulphocyanate may be determined in such solutions by the usual methods, which are not especially difficult where laboratory facilities and experience are available. Wherever possible, it is desirable to have such analyses made at intervals, although by observance of the above suggestions it is believed that the solutions will change in composition less rapidly than under the usual conditions.

5. APPLICATIONS OF "BLACK NICKEL"

In the application of "black nickel" plating to brass, a preliminary copper plating is frequently recommended or specified. It has not been found that deposits produced under such conditions are intrinsically more adherent or serviceable than those produced directly on brass. Where, however, the "Government bronze" effect is desired, the copper plating is an advantage in that whenever the black coating is removed—for example, by abrasion—an underlying copper coating produces a more pleasing effect than does brass. If the copper plating is so produced as to have a dull surface, it also tends to produce a matte effect in the "black nickel." This is frequently desired, especially when the sulphocyanate solution is used, which, on a polished surface, tends to produce a bright black finish. The actual wearing properties of the "black nickel" surface are largely determined by the quality of the lacquer subsequently applied.

Although "black nickel" may be deposited directly upon steel, it is more difficult to secure good results than if the steel is first plated with some other metal. For this preliminary coating, copper and nickel are most frequently used. Since, however, neither the copper nor the nickel, nor the "black nickel" deposits afford any appreciable protection against corrosion, finishes so

produced are not desirable for outdoor exposure. For this purpose the steel should be first zinc plated, either in the sulphate or cyanide bath. Experiments have shown that steel plated with zinc and then with "black nickel" possesses about the same resistance to the salt spray test as does steel plated with the same thickness of zinc. In other words, the "black nickel" does not appreciably add to or detract from the protective value of the zinc coating.

6. CONCLUSION

A bath prepared from the ingredients here listed, kept neutral with an excess of suspended zinc carbonate, and operated with nickel anodes will give satisfactory black deposits upon brass, copper, and zinc; upon brass plated with copper; and upon steel plated with copper, nickel, or zinc. The deposit is probably a mixture of nickel, nickel sulphide, zinc, and organic matter of undetermined composition.

Ingredients	Ounces per gallon	Grams per liter
Nickel ammonium sulphate	8	60
Zinc sulphate	1	7.5
Sodium sulphocyanate	2	15

The authors desire to acknowledge the assistance of L. M. Ritchie and A. D. Bell in certain of the tests, and that of Dr. William Blum for suggestions and advice during the investigation.

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No. 191

SOME FACTORS AFFECTING THE LIFE OF MACHINE-GUN BARRELS

BY

W. W. SVESHNIKOFF, Assistant Physicist

Bureau of Standards

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SOME FACTORS AFFECTING THE LIFE OF MACHINE-GUN BARRELS

By W. W. Sveshnikoff

ABSTRACT

Star-gage measurements on six machine-gun barrels at various stages of firing indicate that when the life limit is reached, exhaustion is due to a combination of the abrasive action of the bullet and the abrasion by hot gases.

In this experiment the amorphous martensite which appears on the surface of the bore of the fired gun has been reproduced on machine-gun steel by the heat from an electric arc, the formation of the martensite being due to the extremely rapid cooling caused by the large mass of cold metal near the highly heated surface.

Cracking of the bore is due to dimensional changes of the hardened brittle surface of the steel resulting from the sudden changes in temperature between separate shots. The cracks originate at irregularities in the surface of the bore.

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I. INTRODUCTION

It is the object of this investigation to study some of the principal factors which affect the life of a machine gun. Previous investigations have been confined to a study of the life of large guns, where firing conditions are different from those of machine guns. The great value of the latter lies in the accuracy of its aim and its rapidity of fire, which are to a great extent dependent upon the construction of the barrel.

The firing of the gun causes the wearing away and surface cracking or the bore, with its gradual enlargement from the breech end.

By this means the rifling is destroyed and the projectile loses its velocity and proper speed of rotation, thus causing inaccuracy in flight. At this stage the barrel is considered useless, especially for barrage firing.

The life of a machine-gun barrel is numerically measured by the number of rounds which can be fired before the barrel loses its accuracy. It is difficult to predict accurately the life of such barrels, as it has been found that there is considerable variation

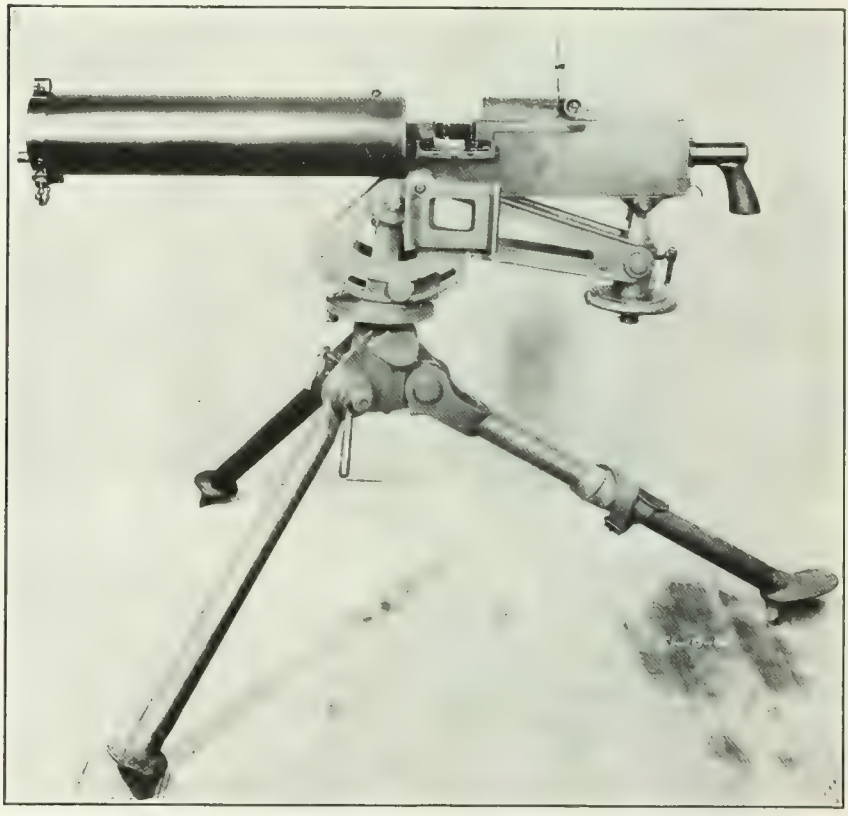

FIG. 1.—*Typical machine gun (Browning 1917 Model)*

in similar barrels in the number of rounds which can be fired before the barrel is reduced to scrap metal. The life of the machine-gun barrel depends to some extent on some other causes not discussed in this paper, for example: The rate of firing, the number of rounds fired in each burst, the time intervals between bursts, the effectiveness of the water cooling, the type of ammunition used, the cleaning of the barrels, and the atmospheric conditions and temperatures. The effect of temperature and pressure for one

shot during the fraction of a second required for the bullet to travel the whole length of the barrel may be described as follows: The combustion of the charge creates an extremely high temperature (about 2800° C) with a pressure of about 51 000 pounds per square inch in the breech end. The effect of this high temperature and pressure is greatest near the breech end of the barrel. The pressure and temperature are respectively decreased with forward movement of the bullet because of the increase in available volume of the barrel and the greater heat absorption by the larger exposed surface of the bore.

II. DISCUSSION OF THE RELATIVE ABRASIVE ACTION OF THE BULLET AND THE GASES

Relative variations in the diameters of grooves and lands as the firing of a machine gun progresses are shown in Fig. 2.[1] The curves represent measurements made with star gages on six barrels and are accurate within a few ten-thousandths of an inch.[2] From a comparison of the average diameter of the bullets and the average diameter of the bore in the grooves, it would appear that the bullet is not in contact with the grooves in a majority of cases. Possibly the bullet is slightly enlarged in girth due to the deformation to which it is subjected upon entering the rifling. Consequently, the base of the bullet will in early firing come into contact with the grooves, and at the same time the bullet (being of greater diameter than the diameter of the lands) is in contact with the lands, which results in considerable friction. Star-gage measurements show that after firing about 2000 rounds the grooves at the breech end are eroded to such an extent that the surface of the bullet no longer makes a sealing contact. At the distance of about 21 inches from the muzzle, as shown in Fig. 2, the increase in diameter along the lands is greater than that in the grooves. At about 15 inches from the muzzle and from this point in the direction of the muzzle the increase in diameter in the grooves is greater than on the lands. A continuous increase in diameter on the lands and grooves from the muzzle to the breech ceases at a distance of about 9 inches from the muzzle. This, it is thought, may be due to deposition of volatilized material at this section of the barrel.

[1] Curves plotted from data obtained from the Ordnance Department, U. S. Army.

[2] The diameter in the lands and consequently in the grooves varies slightly for the separate guns. However, the difference between the two diameters as they increase after a certain number of rounds is practically constant for various guns. The increase in diameters is the same for guns which originally had the same diameters both in grooves and in the lands.

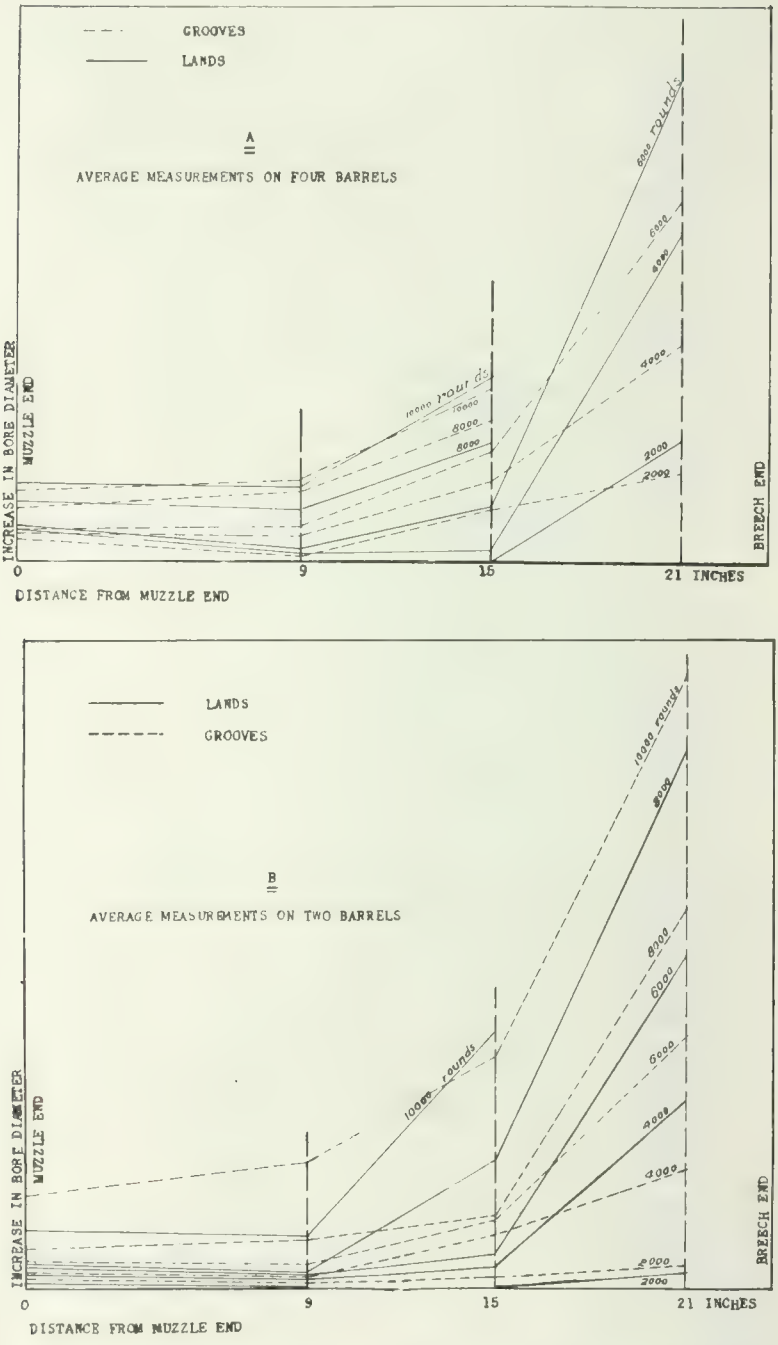

FIG. 2.—*Graphs showing relative enlargements of lands and grooves of machine-gun barrels with increased number of rounds fired. Star gage measurements were taken at four different distances from the muzzle end*

Relative increases only are available for publication

The increase in the diameter on the lands is greatest in the breech end and is very slight at the muzzle. This may be explained by (1) the higher temperature at the breech near the burning powder, and (2) the longer interval of time for which this part of the barrel is exposed to the high temperature. This second conclusion is based on Earle's [3] assertion that about 50 per cent of total time of travel of a projectile from origin of rifling to the muzzle is spent in traveling one caliber down the bore. A similar condition obtains to a smaller degree in the machine-gun barrel. The greatest abrasion is naturally at the breech end, where the bullet enters the rifling. The diameter of the lands is gradually increased as successive bullets enter the rifling, and the zone of the greatest abrasion due to this action is extended for a short distance down the bore.

The increase in the diameter in the grooves is greater also at the breech and diminishes toward the muzzle. Higher temperature and pressure at the breech end, as previously explained, are responsible for the greater abrasive action of the overheated gases.

The hot stream of gases rushes between the bullet and the walls of the grooves with great velocity. There is slight contact between the bullet and the grooves, and each shot increases the average diameter of the latter by melting or possibly by vaporizing the metal. This washes the grooves throughout the entire length of the barrel. With the resulting enlarged free space between the grooves and the moving bullet, the abrasive action of the gases is increased because of the larger volume of gases escaping through the free space.

It will be noted in Fig. 2 that the differences in the diameter on the lands after 2000 rounds have been fired is negligible throughout the first 15 inches from the muzzle, while the diameter in the grooves increases considerably in comparison throughout this entire distance. It is also shown that with an increased number of rounds the increase in the diameter of the grooves is greater than the increase in the diameter of the lands. If the wearing away of the surface of the bore by the gases were the same on the lands and in the grooves, the increase in diameter of the former should be at least equal to the increase in the diameter of the latter for the same number of rounds. But the grooves show a greater increase in diameter than the lands, which suggests that the greater erosion in the grooves, over this distance (15 inches from

[3] Ralph Earle, Transactions of the American Institute of Mining Engineers, **56**, p. 495, 1917.

muzzle) is mainly due to the escape of highly heated gases around the seat of the bullet.

Consequently when a machine-gun barrel reaches its life limit its exhaustion is due to a combination of the abrasive action of the bullet and abrasion of gases, but to a greater degree to the former. The action of the bullet obliterates the lands, which results in the destruction of the rifling, beginning at the breech end. The destruction of the rifling causes the loss of the gun's accuracy, thus terminating its usefulness.

III. THE STRUCTURE OF THE SURFACE LAYER IN THE BORE OF THE FIRED GUN

1. EFFECT OF TEMPERATURE ON THE SURFACE OF THE BORE

The temperature of the surface of the bore rises rapidly with the increased number of rounds fired. The effect of the high temperature penetrates only to a very slight depth, as the gun is rapidly cooled by the large mass of metal of relatively low temperature adjoining the highly heated layer. In the case of machine guns, the combustion is not completed at the time the bullet leaves the muzzle, and consequently, because of the rapidity of the fire, cold air has no opportunity to enter the barrel and exert a cooling effect. In the case of intermittent firing with the machine gun, air enters the barrel and naturally some cooling is obtained. Below the heated surface of the bore the steel has only a moderate temperature. After a limited number of rounds has been fired, only the thin surface of the bore is injured. This is shown in a microscopic examination of the original cross sections, Figs. 3, 4, and 5. The thin structurally changed surface layer

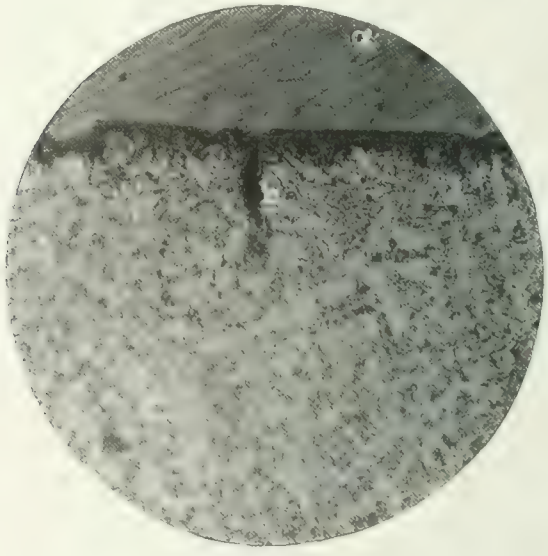

FIG. 3.—*Cross section of a gun barrel showing the thin structurally changed surface layer caused by firing 3880 rounds* × 200

(*a*) Electrolytic protective layer of copper.
(*b*) Highly heated layer;
(*c*) Original structure of steel

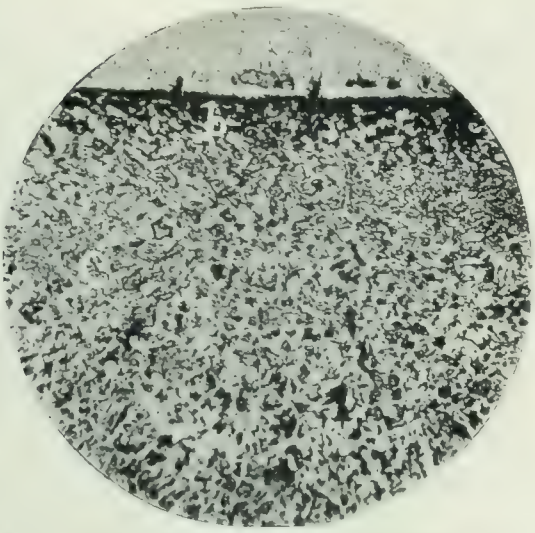

FIG. 4. *Cross section of a gun barrel showing the thin structurally changed surface layer caused by firing 8000 rounds.* × *250*

(*a*) Electrolytic protective layer of copper.
(*b*) Highly heated layer;
(*c*) Original structure of steel

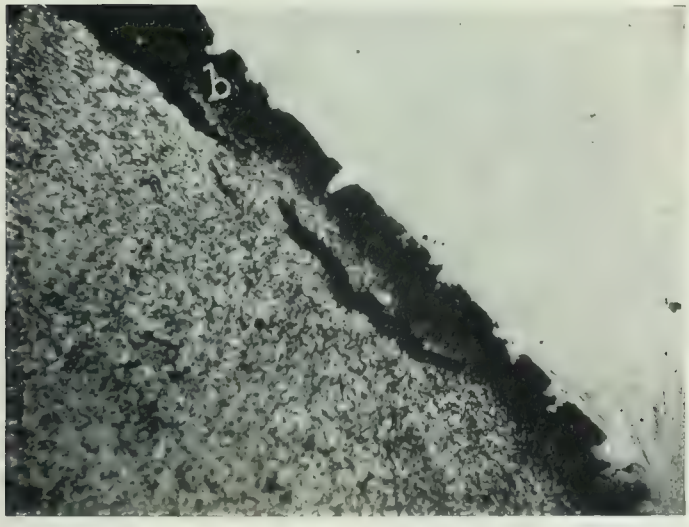

FIG. 5.—*Cross section of a machine gun barrel after firing 8000 rounds* × *250*

(*a*) Electrolytic protective layer of copper
(*b*) Highly heated layer.
(*c*) Thin white layer

41566°—21——2

originates at the breech end of the barrels, where the duration of heating has been longest and the temperature highest. The depth and length of this surface layer depends upon the temperature of heating, and it grows toward the muzzle with continual firing. For a gun fired 8000 rounds the depth of this layer at the breech is about 0.0013 inch.

Fig. 5 also shows the appearance of the thin white layer on the surface of the bore, but because sufficient information concerning it is not now available its nature and structure are not discussed in the present paper.

2. EXPERIMENTS AT THE BUREAU OF STANDARDS ON THE INTENTIONAL PRODUCTION OF THE HEATING EFFECT BY AN ELECTRIC ARC

In studying this phase of the problem experiments were carried out with intentional production of this heating effect on the steel by an electric arc.

The temperature of combustion of nitrocellulose powder is estimated to be about 2800° C. The temperature of the hottest part of the arc from a positive carbon electrode is estimated by C. W. Waidner and G. K. Burgess [4] to be between 3600 and 4700° C, which is considerably higher than the temperature of combustion of powder. In either case a temperature higher than the melting point of steel is obtained. The temperature attained by the metal of the bore increases with an increase in available heat. The available heat is increased by an increased rate of firing. A prolonged exposure of the surface of the gun to this heat would even result in fusion of the surface layers.

According to Prof. Zay Jeffries [5] depression of the Ac^1 transformation point under conditions existing within a gun at the time of firing should be slight. The change of melting point due to the increased pressure within the gun is so small as to be negligible.

Machine-gun barrel steel stock used for the arcing experiments conforms to the following chemical specification:

	Per cent
Carbon	0.40 to 0.50
Manganese	.60 to .70
Phosphorus	Maximum, .045
Sulphur	Maximum, .035
Silicon	0.20 to .30
Chromium	.40 to .50

Figs. 6, 7, and 8 show the cracking produced in the heated surface by arcing with a copper electrode. The structure of the

[4] C. W. Waidner and G. K. Burgess, The temperature of the arc, B. S. Sci. Papers, No. 8; Sept. 1, 1904.
[5] Zay Jeffries, Trans. Am. Inst. Min. Eng., 58, p. 581; 1918.

FIG. 6.—*The cracks produced in the surface of steel by the heat of an electric arc (a copper electrode was used). Etched with 2 per cent nitric acid.* × *100*

The dark central part is the hole made by the arc. The white portion of the specimen immediately surrounding the cracks is metallic copper. The structure of the specimen in the area around the cracks is entirely martensitic, merging rather abruptly with the original ferrite sorbite structure of the specimen

FIG. 7.—*Same surface as in Fig. 6, but after continued polishing of sample. Etched with two per cent nitric acid.* × *100*

specimen in the area around the crack is entirely martensitic and merges rather abruptly with the original ferrite-sorbite structure in the specimen. Figs. 6 and 8 also show main and tributary cracks following the grain boundaries. The white portion of the specimen immediately surrounding the crack is metallic copper which has been deposited by the electrode during the arcing. A flow of copper has also occurred along the grain boundaries which are brought out by the high temperature.

To avoid the copper deposit, a specimen of this machine-gun steel was exposed to the heat of an electric arc drawn from a carbon electrode. Fig. 9 shows main and tributary cracks in the surface of the specimen caused by this method of arcing. During this test a reddish brown deposit, which probably consists of iron and manganese oxides, was formed on the surface adjacent to the heated area. This experiment was repeated several times, and in each case a deposit of iron, spongy in appearance and devoid of structure, was apparent at the point of arcing. The deposit was filled with inclusions, probably oxides, while the structure of the surrounding metal appeared to be martensitic, merging sharply with the original sorbitic ground mass.

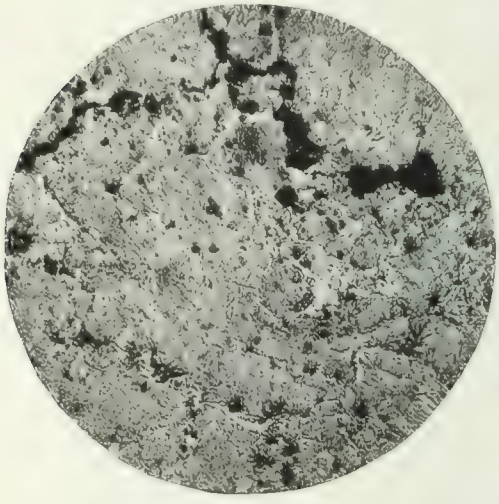

FIG. 8.—*Micrograph showing cracking in the specimen submitted to electric arc from the copper electrode. Grain boundaries developed. Molten copper tends to follow grain boundaries. Indicative of the origin of these boundaries while the copper was still plastic and capable of flowing. Etched with 2 per cent nitric acid.* × 100

Fig. 10 shows the dark grain outline in the martensite bordering the edge of the view shown in Fig. 7.

In the specimen of machine-gun barrel steel examined which had been submitted to arcs from the copper and carbon electrodes, the occurrence of grain boundaries is significant of the high temperature to which the metal has been heated. The grain boundaries are best shown in Figs. 6 and 8. They are well developed at a short distance from the fused area, and they end abruptly just within the area which had been changed to marten-

FIG. 9.—*Micrograph showing main and tributary cracks produced in the surface of steel by the heat of an electric arc (a carbon electrode was used). Etched with 2 per cent HNO_3 in alcohol.* × 400

The microstructure of the area surrounding the crack is martensitic, merging sharply with the original sorbitic ground mass

site. In the specimen heated by the copper electrode the boundaries adjacent to the fused area are filled with copper, but in the specimen heated by the carbon electrode fused iron has filled in the cracks for a shorter distance.

The micrographs show that the metal had been exposed to a very high temperature, permitting volatilization of the amorphous metal between the grains.[6] The amorphous phase has a higher solution pressure than the crystalline phase, and although volatilization would probably occur in both of these phases, the greater loss would occur in the amorphous phase. After volatilization had proceeded along the grain boundaries for a slight

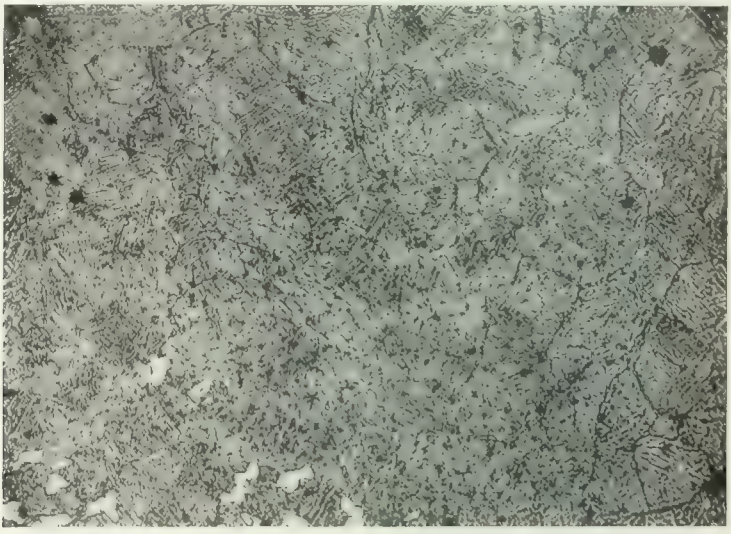

FIG. 10.—*Micrograph showing development and termination of grain boundaries in the martensitic mass. Section taken adjacent to sorbite-ferrite ground mass of unaltered area of the specimen. Etched with* HNO_3. × 500

depth, the pool of molten metal gradually followed the depressions which had been formed.

The conditions of the change in structure and the grain-boundary development are direct results produced by the high temperature. The pool of molten copper could not have been deposited under a temperature of 1075° C. The steel fused in the case of an arc from the carbon electrode would not have melted under a temperature of 1500° C

In the micrograph shown in Figs. 6 and 8 the branches of the crack appear to show a preference for the grain boundaries. This

[6] W. Rosenhain and D. Ewen, Intercrystalline cohesion in metals, J. Inst. of Metals, 8, p. 149; 1912

would also point toward the high-temperature origin of these cracks. It is known that fracture produced in steels at high temperatures is intercrystalline.[7] It has further been demonstrated that the cohesion between adjacent crystals of iron at temperatures above about 650° C is not as great as the cohesion between units of the same crystals.[8] Furthermore, the maximum expansion or contraction occurs in steel passing through the Ar_1 or Ac_1 ranges. At and above these temperatures the metal lacks sufficient mobility to adjust itself to the rapid changes in volume; consequently internal stresses begin their attack on the weaker portion of the metal; that is, the grain boundaries. By the time a temperature of 550° C has been reached in cooling, most of the stresses incident to contraction and expansion have expended their energy and caused the appearance of an intercrystalline fracture. Below that temperature the path of a crack caused by tensional stresses should pass indiscriminately through the metal without regard to its relative crystalline or amorphous condition.

Belaiew and Rosenhain [9] reported that the thin surface layer in the bore of a fired gun consisted of martensite, and the investigations of Prof. H. Fay [10] confirm their report. Prof. Fay carried out his experiments with pressure plugs, previously subjected to cold work, which were inserted in a gun later fired. In the examination of these plugs he found the effect of hardening with cracking in the vicinity of the cold-worked areas. This indicated that the hardness of the surface was due to a combination of mechanical deformation and a process of martensitization. Prof. Fay considered these cracks to have appeared before the change of structure had taken place. Further, the structure, in his opinion, when it first changes appears as troostite, which develops into amorphous martensite. Prof. Fay's observation of a change from the sorbitic to the troostitic structure in the case mentioned did not occur in the specimen of machine-gun steel examined.

It is difficult to conceive of a troostitic structure forming from a sorbitic ground mass, because such transformation occurs in the direction which is the reverse of the usual change. However, the sorbitic ground mass may be transformed into martensite, which subsequently breaks down and is caught at the troostite transition

[7] J. C. Humfrey, Influence of intercrystalline cohesion upon the mechanical properties of metals, Iron and Steel Institute, Carnegie Scholarship Memoirs, 5, p. 85, 1913.

[8] Zay Jeffries, The amorphous metal hypothesis and equicohesive temperature, J. Am. Inst. of Metals, 11, p. 486, 1917.

[9] Belaiew, Rosenham, International Association for Testing Materials, Proceedings, 2, Sec. A; 1912.

[10] Prof. H. Fay, Trans. Amer. Inst. Min. Eng., 55, p. 468; 1917.

FIG. 11. *Part I (gun barrels Nos. 1, 2, and 3). Longitudinal sections of machine-gun barrels after firing different numbers of rounds*

(*a*) Cracks in lands
(*b*) Cracks in grooves.
(*c*) Juncture cracks

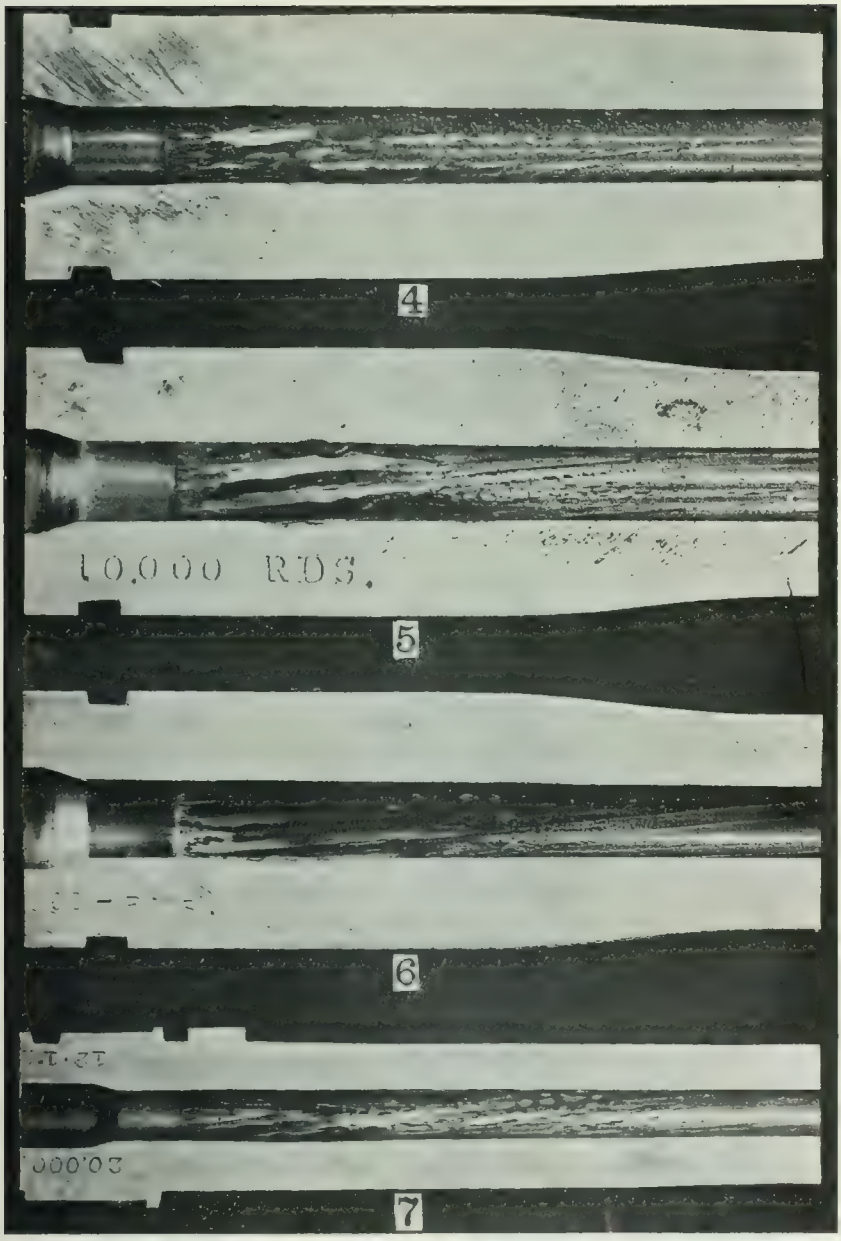

FIG. 11.—*Part 2 (gun barrels Nos. 4, 5, 6, and 7). Longitudinal sections of machine-gun barrels after firing different numbers of rounds*

(*a*) Cracks in lands;
(*b*) Cracks in grooves;
(*c*) Juncture cracks

These photographs were obtained from the Technical Staff of the Ordnance Department, U. S. Army.

point. Troostite is a decomposition product of martensite. It usually forms on tempering a martensitic steel, but it can be formed directly by quenching. In either case the martensite-troostite-sorbite cycle appears to favor the formation of martensite before troostite can occur. The occurrence of the amorphous martensite in the surface layer of the gun can be conceded.

IV. CRACKING OF THE BORE OF MACHINE-GUN BARRELS

1. VISUAL EXAMINATION OF CRACKING

Firing of a machine gun causes fine cracks to appear in the surface of the bore. These cracks progressively increase in length, depth, and width with the number of rounds fired, as shown in Fig. 11, which growth is accompanied by increased area of the surface of the bore being exposed to the high temperature. In Fig. 3 the effect of heating upon the walls of the cracks will be observed.

From a study of the cracks in the bores of a number of machine-gun barrels it was found that the cracks are better developed and more numerous at the breech end, as this is the point where the temperature is at a maximum. From the examination of longitudinal sections of gun barrels fired it was found that the effect of cracking was at a maximum at a short distance from the origin of rifling and not at the origin, as one would expect. As firing is carried further, the bore is more rapidly destroyed at this point than at other portions of the barrel. It is believed that this condition is due to the fact that the stream of hot gases leaving the cartridge is partially deflected by the shoulder and neck of the shell.[11] Thus it is only natural that they are directed toward a point at some distance from the origin of rifling where a maximum temperature is reached. This effect is shown in Fig. 11, barrels 4, 5, and 6.

The cracks decrease progressively in depth and extend from the breech toward the muzzle. The majority run spiral-longitudinally in the grooves and transversely on the lands, as shown in Fig. 11. Cracks at the sharp junction of the lands and grooves usually are more pronounced and continue unbroken throughout the length of the barrel toward the muzzle, and often appear in those sections of the bore beyond a point where other cracks seem to have disappeared. On the other hand, all other cracks having the same longitudinal direction in the grooves are broken or dis-

[11] This refers to the bottle-neck shell ordinarily used.

continuous. The destruction of the lands, that is, lowering caused by the abrasive action of the bullet, is greater than the erosion on the grooves caused by the abrasive action of gases, but the cracking in the lands is somewhat less than that in the grooves. (See Fig. 11, barrels 1, 2, and 3, point "a," lands, and point "b," grooves.)

2. THEORY OF PROF. D. TSCHERNOFF [12]

Various explanations have been offered to account for the development of cracks in the bore of fired gun barrels. An extract from the theory of Prof. Tschernoff,[13] based solely on the physical action of heat, is given below:

The first evidences of erosion are manifested in the appearance of dull spots upon the brilliant surface of the bore, principally in the upper part of the chamber, in the vicinity of the centering cone, and on the cone itself at the origin of the rifling. Examination of gutta-percha impressions shows that these spots are made up on networks of small cracks, extremely fine and very shallow. As a rule, the meshes of these nets are not closed in the incipient stage (Fig. 12 *a*); but as the firing is continued, the little isolated cracks extend, joining the neighboring ones and forming a network of continuous meshes (Fig. 12 *b*).

FIG. 12.—*Sketch illustrating Prof. Tschernoff's theory of the appearance of the cracks in the bore of a gun*

With repeated firing, the little cracks increase in width and depth, but on the centering cone, and at the origin of the rifling, this increase is particularly noticeable in the cracks parallel to the axis (Fig. 12 *c*). The gases of the charge and the unburned particles of powder penetrate between the projectile and the bore and follow the lines of least resistance, cutting them deeper. This deepening of the cracks parallel to the axis of the bore clearly indicates mechanical action of the products of combustion.

On the lands and on the edges of the grooves, transverse cracks or scorings predominate; as soon as a fissure appears in a plane at right angles to the axis of the bore, the rotating band of the projectile enters it, tears away particles of steel, and enlarges it at each shot. The erosion sometimes proceeds so far as to remove the lands completely, or even to make hollows where the lands were.

Tschernoff in his explanation of the formation of transverse cracks in the surfaces of the lands maintains that longitudinal expansion of the lands does not occur, being prevented by the restraining influence of the cooler layers which underlie the highly heated skin, the cooler layers being in turn restrained by their integral union with the body or bulk of the lands.

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[12] Prof. Tschernoff made his observation on the longitudinal sections of two Russian 6-inch pieces which had been fired about 3000 times.

[13] Capt. H. Pelloux, of the French Artillery. Translation of the Tschernoff theory of erosion, Journal of U. S. Artillery, 41, p. 346; 1914.

The lands, however, do have the opportunity to expand laterally, in which direction most of the expansion upon heating occurs. Upon cooling, a longitudinal contraction does not occur on the body of the lands. At the other extreme, the metal at the surface, which has been highly heated, tends to contract equally in both longitudinal and transverse directions, but the great longitudinal stress which is thus set up upon cooling in the surface layer is beyond the elastic limit of the previously heated metal, so that it cracks transversely.

This explanation affords some comfort in endeavoring to describe conditions leading to transverse cracking. However, the greater lateral expansion which the author mentions should be accounted for. If the explanation of transverse expansion accounts for the longitudinal stresses, a similar explanation should suffice to account for the lateral or transverse stresses. These, he contends, are held within the elastic limit.

However, the surface metal heated to the temperature which his explanation requires would have expanded in all directions so that parallel conditions should apply in the formation of longitudinal cracks. The absence of pronounced longitudinal cracks, and the fact that the majority of the cracks in the lands are transverse, appears to make Tschernoff's explanation partially untenable.

3. RELATION BETWEEN THE DIRECTION OF THE CRACKS AND THE MACHINE MARKS

Differences in the direction of the cracks in the lands and the grooves were noted in machine-gun barrels which had been fired for different numbers of rounds. The same differences in the direction of the cracks for the guns of large calibers were reported by several investigators. It was apparent that the cracks and the machine marks from tools were invariably in the same direction. The machine marks in the unfired barrel are also transverse in the land and of spiral-longitudinal direction in the grooves, as illustrated in Fig. 13.

This is due to the method of rifle manufacture, as may be seen from the following:

The barrel blank is drilled first with a rotating motion perpendicular to the long axis, which operation is followed by reaming The motion of the reamer coincides with that used in drilling. The lands are made in this manner, and naturally from the rotation of the machine tools, marks from machining are transverse

to the long dimension of the land. The grooves are made by moving the rifling head of the scrape cutter on a spiral curve. The operation produces machine marks which run in the direction of rifling, which is spiral longitudinally.

This similarity in the direction of the cracks and the machine marks in the grooves and lands would appear to indicate that these marks are the starting points of the heat-contraction cracks. Naturally, if one fires from a machine-gun barrel with an absolutely even bore, without rifling and with a perfectly polished surface, cracks, if any, should appear and develop equally on all parts of the surface. On the other hand, if the barrel has a rifled surface or any marks on the bore, cracks would be expected to originate at these irregularities, such as the corners of the grooves, and later at all parts of the bore which in the beginning had uneven surfaces.

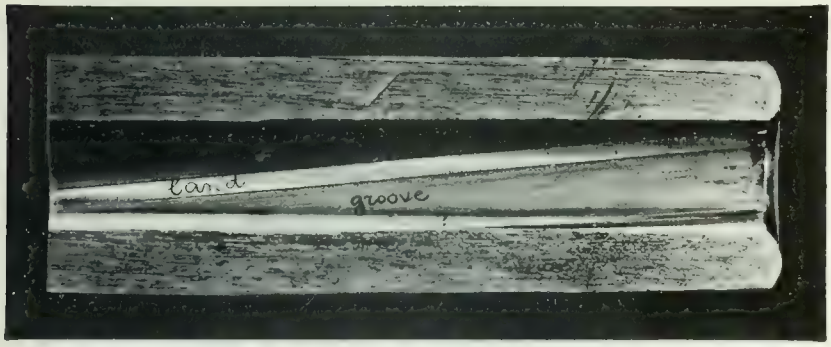

FIG. 13.—*Interior view of the gun bore, showing the direction of marks of the cutter of the machine tool in the grooves and the lands of the unfired barrel*

The most marked of these irregularities naturally would be the points at which cracks originate, as it is a fact that steel ruptures quickly where there are any sharp changes of direction in its surface.

The cracks in the grooves and the lands follow the path of the machine marks. Once begun they move in the direction of least resistance, following the direction of the marks made by the machine tool in the already defective surface. Each shot naturally causes the crack to grow larger and deeper. It has been found that the cracks in the machine-gun barrel produced by actual firing are sharp ended, as shown in Figs. 14 and 15. Prof. Howe [14] stated in regard to the sharp-ended cracks in cannon barrels, that if they were deepened by abrasions of gases, they would be

[14] Prof. Howe, The erosion of guns, Trans. Amer. Inst. Min. Eng., 58, p. 542; 1918.

FIG. 14. *Cross section of a machine-gun barrel fired 3880 rounds, showing cracks running radially from the bore. Specimen was taken at origin of rifling.* × 8.8

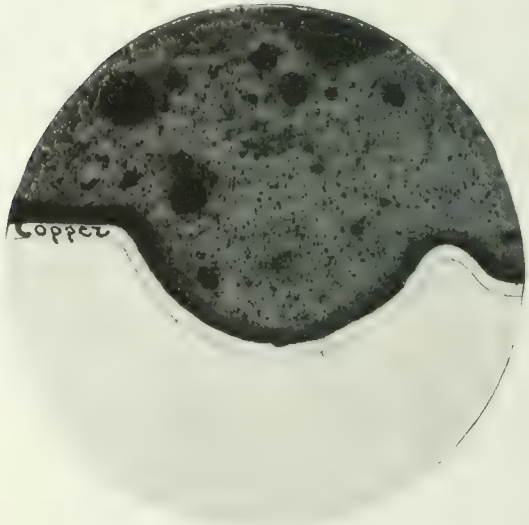

FIG. 15.—*Cross section of a machine-gun barrel fired 3880 rounds, showing cracks running radially from the bore.* × 6.2

Specimen was taken about 4 in. from the origin of rifling.

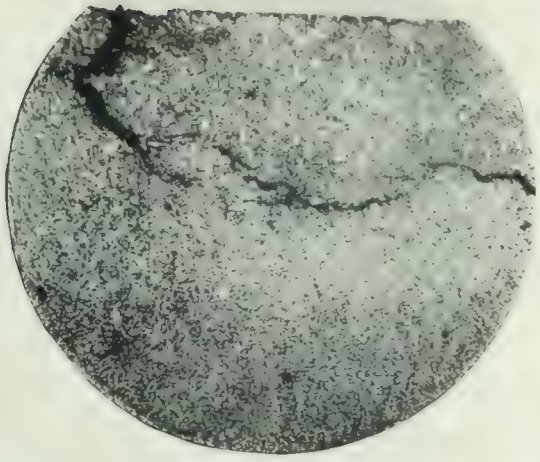

FIG. 16.—*Cross section of a machine-gun barrel after firing 4000 rounds, showing circumferential and radial cracks.* × 250

(*a*) Electrolytic protective layer of copper

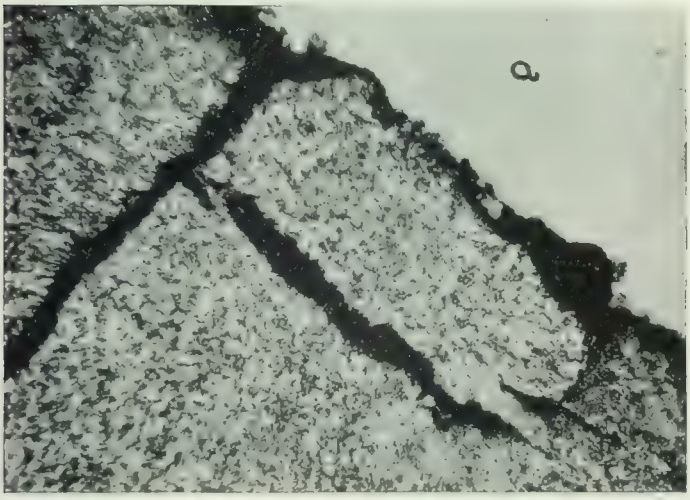

FIG. 17.—*View similar to that of Fig. 16 after firing 8000 rounds.* × 250

Note complete connection of circumferential cracks, which will cause the separation of the piece on continued firing

(*a*) Electrolytic protective layer of copper

round bottomed, and that the spindling of the cracks showed that they were opened mechanically. The heat cracks produced in the steel by arcing, which extend into the body of the metal, are sharp ended, as is shown on the micrographs, Figs. 6 and 9.

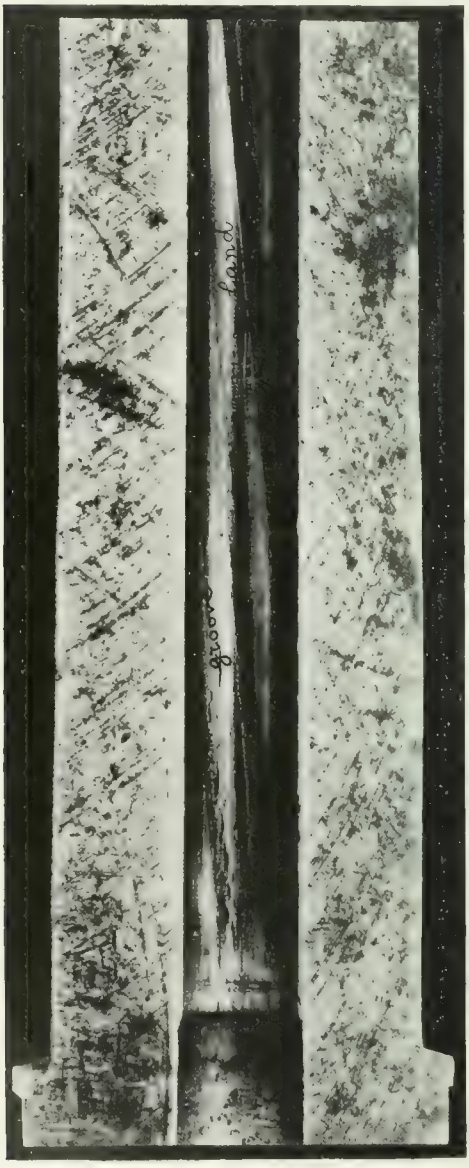

FIG. 18.—*Interior view of gun bore showing cracks agreeing in direction with the marks of the cutter of the machine tool. Gun fired 3880 rounds.* × 2.3

Besides the radial cracks, Figs. 14 and 15, which start from the surface of the land and groove proceeding outward, cracks also form circumferentially. Circumferential cracks may originate (1) at the junction of highly heated metal with metal heated only to a moderate temperature, or (2) at any point subject to the tensional stresses created in the expansion and contraction of the steel. The crack illustrating (2) is shown in the micrographs, Figs. 5, 16, and 17. Circumferential cracks would connect the radial cracks together, thus helping in the destruction of the rifling, with an enlargement of the bore resulting.

In an effort to obtain a very high finish in manufacture, "lapping" the barrels after rifling was tried by moving a lead core which fits the lands and grooves through a bath of flour emery and oil. The "lapping," which follows the rifling in its spiral

motion, does not completely remove the machine tool marks in the grooves, as it is in the same direction. However, moving in a direction perpendicular to the machine marks made by drilling in the lands gives a better finish. The "lapping" of the barrels, as is the practice in manufacture, fails to increase the life of the guns. It is believed, however, that by an extremely long "lapping," giving a higher polish to the surface, the life of the barrel would be increased.

The transverse cracks in the lands appear and extend in the same direction as the transverse machine marks: the scoring action of the bullet causes the loss of their transverse direction with increased number of shots. In the grooves, however, the cracks remain distinctly longitudinal, agreeing with the direction of the machine marks, as shown in Fig. 18. Forming in the machine tool marks, the cracks themselves became connected, following the outlines which already exist between the marks.

4. COMPARISON OF CRACKS IN THE LANDS AND THE GROOVES

From the above it is understood that most marked irregularities occur at the juncture of the lands and grooves, and these irregularities produce cracks which are more pronounced than any of the other cracks in the grooves (Fig. 11, points "c"). These cracks can be observed even when the rifling of the barrel is spent, and they clearly indicate the former position of the lands.

The explanation of Prof. Tschernoff [15] may also be of interest in regard to the origin of these particular cracks. He states that the contraction and expansion act differently in the grooves than in the lands in accordance with their design, the lands being heated from three sides, while the grooves are heated from one side only. The difference in contraction and expansion of these two adjoining parts might increase the effect of cracking and help the formation of the most pronounced cracks at these particular points.

The cracks in the lands probably develop at the same time as in the grooves. Difference in the extent of cracking in the grooves, where cracks are more pronounced than in the lands, is due to the differences in the abrasive action of the bullet. The motion of the bullet is perpendicular in its direction to the machine-tool marks in the lands; in the grooves, however, the motion of the bullet is in the same direction with the machine-cutting marks. As previously stated, the bullet is in slight contact with the grooves

[15] Prof. Tschernoff, The Artillery Journal of Russia, 7; 1912.

only for a limited number of rounds. The lands, however, are under considerable friction. One would hardly expect the scoring-polishing effect in the grooves to be equal to that in the lands. As a result of these differences in abrasion, cracks in the lands are reduced in their lateral dimensions by the wearing away of their upper walls. Cracks in the grooves are not exposed to the abrasive action to the same extent, consequently they are able to develop greater lateral openings. If the machine-gun barrels are not fired beyond the complete obliteration of the rifling, the difference in the color of the lands and the grooves may be noted. Hence the bullet lowers the lands with each passage but scores (polishes) the grooves at the same time to a much lesser extent, and only during a limited number of rounds. Hence the formation and growth of cracks in the grooves is not retarded as in the lands by the abrasive action of the bullet.

It is believed that a slight rounding of the sharp junction between the lands and the grooves as well as the upper corners of the lands in the rifling should decrease the susceptibility to cracking in these particular points and should prolong the life of the rifling for the reasons previously outlined.

V. SUMMARY.

This investigation indicates that when a machine-gun barrel reaches its life limit its exhaustion is due to a combination of the abrasive action of the bullet and abrasion by hot gases, but to a greater degree to the former.

The author's experiments using the electric arc show that the rapid cooling (which is due to the large mass of cold metal near the highly heated inner surface of the steel) from temperatures near the melting point of the metal produces a martensitic layer. A similar layer is produced in the firing of a machine gun, indicating that the temperature conditions for the development of martensite can be made by the electric arc to approach those which occur in the gun under actual fire.

The selective hardening of the steel sets up surface strains, and the surface of the bore is readily cracked on account of the dimensional changes of the hardened brittle surface of the steel resulting from sudden changes in temperature between separate shots. The cracks that originate at irregularities in the surface of the bore, are attributable to the method of manufacture of the barrels.

The work herein reported was carried out under the direction of G. K. Burgess, chief, Division of Metallurgy of the National Bureau of Standards. J. S. Vanick, research operator of the Ordnance Department, gave valuable assistance in the microscopical examination of the specimens and the duplication of arc experiments. H. Fay, Massachusetts Institute of Technology, S. Tour of the Technical Staff, and Capt. S. G. Green of Small Arms Division of the Ordnance Department, U. S. Army, made criticisms and suggestions.

WASHINGTON, September 10, 1920.

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S. W. STRATTON, DIRECTOR

No. 192

TESTS OF CENTRIFUGALLY CAST STEEL

BY

GEORGE K. BURGESS, Physicist
Bureau of Standards

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TESTS OF CENTRIFUGALLY CAST STEEL

By George K. Burgess

ABSTRACT

Six castings manufactured by the Millspaugh centrifugal process were examined as to their physical and chemical properties, including hardness, tensile strength, impact resistance, density, internal stress, segregation, soundness, and microstructure, both in the condition as cast and after various heat treatments. This investigation shows the possibilities of substituting heat treatment for forging in this type of casting.

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I. INTRODUCTION

In 1918 the Bureau of Standards had occasion to examine several hollow steel cylinders made under the direction of W. H. Millspaugh by his centrifugal casting process. The manufacture of these centrifugal castings was carried out under somewhat adverse conditions in two steel foundries which, of course, due to the limited experimental nature of the trials, could not have developed the refined technique that practice would give, nor were efforts made to produce high-grade steel for these tests.

The cylinders were cast in a machine revolving about its horizontal axis. The outer surface of the castings, of walls from 3½ inches to ½ inch thick, were thus against the mold and turning faster than the free inner, cylindrical surface, which was last to freeze. The outer surfaces were fairly smooth, but the interior surfaces were rough.

The results, comparing the several grades of steel in the condition as cast and after heat treatment, are of considerable interest as indicating what may be expected from this method in the

production, for certain shapes, of sound steel, that is, free from physical defects and chemical segregation, and thus with practically no waste material to discard. As will be shown, it is also a field particularly adapted to the elimination of forging and boring operations and the substitution for the former of heat treatment to produce the desired characteristics in the resulting product.

II. DIMENSIONS OF CASTINGS AND OF TEST SAMPLES

In all, six castings were examined, samples from which were subjected to the usual mechanical tests, both transverse and longitudinal, in the condition as cast and after various heat treatments. The characteristics of the metal with respect to internal stress, density, soundness, segregation, and microstructure were also studied

In Figs. 1 and 2 are shown the dimensions of the castings and location of samples for test, except for No. 7, a cylindrical ring 12 inches in diameter and having a wall thickness of ½ inch, from which longitudinal flat tensile bars 12 inches long by 1 inch by $\frac{9}{32}$ inch were taken. The tensile bars from all other castings were standard 2-inch round bars. Samples for chemical analysis were taken in ring turnings for Nos. 4, 5, and 7, and for Nos. 1, 3, and 6 from longitudinal borings in zones numbered 1 to 5 from outside to inside of the casting (Figs. 1 and 2).

III. TESTS OF CASTINGS

1. HARDNESS AND CHEMICAL SURVEYS

In Table 1 are given the results of the radial surveys for hardness and chemical analysis. It will be noted there is a gradual increase in carbon from the outside to the inside surface for all castings, this increase ranging from 0.02 per cent for No. 3*A* to 0.09 per cent for No. 5; it appears to be roughly proportional to the carbon content, or the percentage variation in carbon remains practically constant. The nickel and phosphorus appear to follow the carbon very closely in their behavior as to segregation; manganese and silicon, on the other hand, are nearly constant across the radial section; while sulphur, although somewhat erratic, in general is distributed similarly to carbon, as is also copper, although present in quantities less than 0.1 per cent. The hardness surveys, Brinell and scleroscope, follow closely the chemical segregation, the higher numbers occurring on the inside layers. A hardness survey with the scleroscope made midway along a longitudinal section the length (72 inches) of casting No. 1 at 3-inch intervals

shows a gradual increase in hardness from end *B* to the middle, 24.2 to 25.3, and then nearly uniform hardness for the second half, or from center to end *A*. (See Fig. 1.) The chemical analysis of

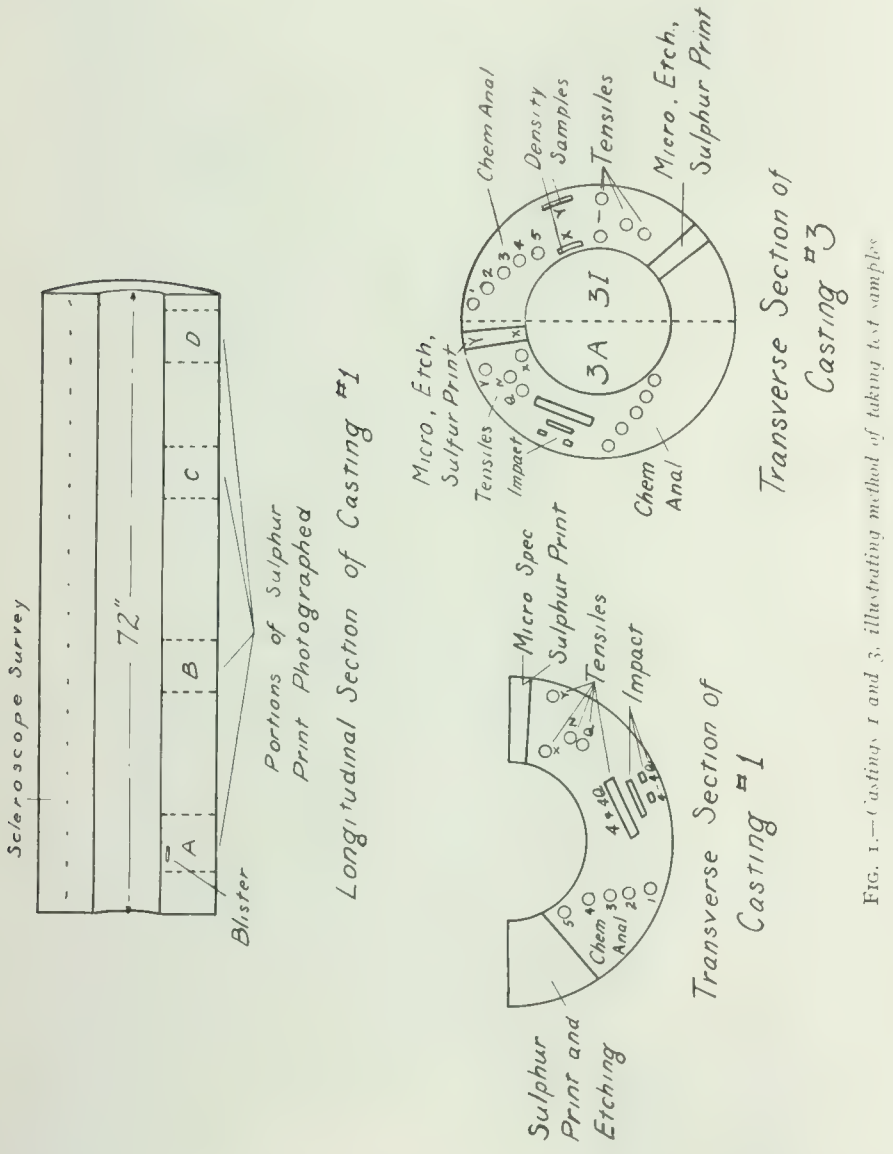


FIG. 1.—Castings 1 and 3, illustrating method of taking test samples

the *A* and *I* sections, separated by 60 inches in casting No. 3, apparently shows evidence of longitudinal segregation, but this is so very slight as to be uncertain.

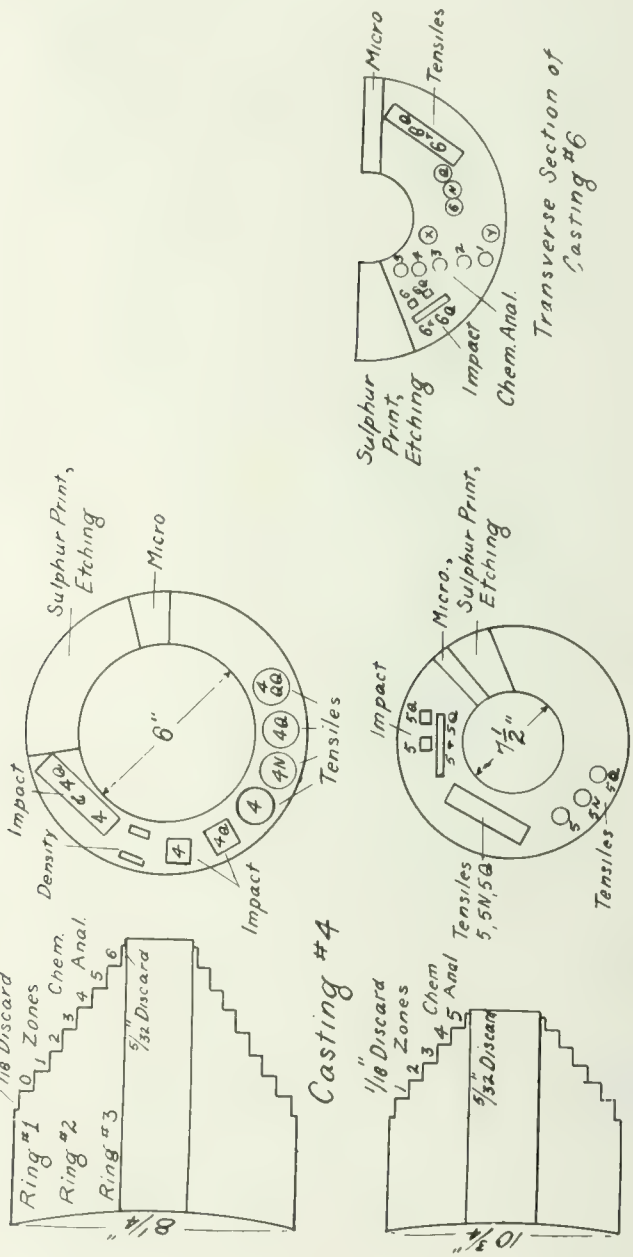


Fig. 3. Castings 4, 5, and 6, illustrating method of taking test samples

TABLE 1.—Results of Hardness and Chemical Surveys

Number	Wall thickness	Zone	Hardness		C	Mn	Si	S	P	Ni	Cu	Cr
			Brinell	Scleroscope								
	Inches											
1	3½	1	186	25.8	0.44	0.44	0.47	0.024	0.011	2.33	0.084	0.04
		2	186	24.9	.48	.44	.47	.030	.013	2.35	.089	
		3	187	25.0	.46	.44	.47	.032	.013	2.32	.085	
		4	187	25.2	.46	.44	.48	.032	.014	2.36	.094	
		5	196	25.7	.50	.44	.47	.034	.015	2.39	.089	
3A	3	1	160	22.3	.33	.54	.19	.022	.015	2.66	.051	.04
		2	164	22.2	.32	.55	.19	.030	.014	2.67	.053	
		3	167	22.4	.32	.55	.19	.032	.015	2.70	.051	
		4	167	22.6	.34	.57	.19	.034	.016	2.75	.055	
		5	160	22.3	.34	.57	.20	.030	.017	2.70	.054	
3I	3	1	158	21.9	.30	.55	.20	.031	.012	2.66	.062	.04
		2	159	22.0	.30	.55	.19	.030	.013	2.67	.062	
		3	158	22.0	.30	.55	.20	.030	.015	2.68	.067	
		4	156	22.5	.34	.57	.20	.030	.017	2.76	.068	
		5	184	23.9	.35	.57	.21	.030	.018	2.60	.065	
4	2¼	0	.....	.....	.16	.47	.27	.054	.045	.....	.....	
		1	125	19.7	.16	.50	.29	.045	.033	.....	.....	
		2	.....	20.4	.16	.51	.28	.045	.033	.....	.....	
		3	128	20.6	.17	.50	.28	.049	.037	.....	.....	
		4	.....	20.0	.18	.50	.29	.053	.038	.....	.....	
		5	121	19.2	.21	.52	.29	.060	.043	.....	.....	
		6	.....	.....	.18	.40	.22	.066	.059	.....	.....	
5	3¼	1	249	63.5	.64	.56	.65	.033	.014	2.94	.....	Tr
		2	.....	60.6	.65	.56	.65	.032	.015	2.92	.....	
		3	248	58.0	.63	.55	.64	.029	.015	2.93	.....	
		4	.....	55.5	.63	.55	.64	.031	.017	2.93	.....	
		5	245	57.0	.72	.60	.68	.033	.017	2.90	.....	
6	3½	1	164	23.4	.32	.56	.19	.034	.015	2.69	.059	.04
		2	166	23.4	.33	.56	.19	.035	.015	2.75	.058	
		3	161	23.4	.31	.56	.19	.039	.015	2.72	.059	
		4	167	23.8	.33	.57	.19	.032	.015	2.76	.064	
		5	191	25.2	.35	.57	.19	.033	.016	2.81	.064	
7	½	1	.....	.....	.22	.54	.26	.026	.013	.....	.....	
		2	123	.....	.21	.53	.25	.026	.014	.....	.....	
		3	.....	.....	.25	.55	.25	.026	.016	.....	.....	

2. DENSITY DETERMINATION

A narrow circumferential strip 2 inches long, 1 inch broad, and ¼ inch thick was cut from the outside and from the inside of a transverse ring from the nickel-steel casting No. 31 and from the low carbon casting No. 4 and used for determination of density. The results are as follows:

Casting No.	Density g/cm³ at 23°C	Difference
3I (outside)	7.838	
3I (inside)	7.834	+0.004
4 (outside)	7.834	
4 (inside)[a]	7.726	+0.108

[a] The low density of sample No. 4 (inside) is probably due to the presence of a series of very fine pits or blowholes in this sample. These holes occur about one-sixteenth inch from the inner surface of the casting, where their presence was actually observed

3. INVESTIGATION OF STRESSES ACROSS SECTION OF RINGS

Three concentric rings, three-eighths inch wide (radially) and one-half inch thick (longitudinally), were cut from the No. 4 casting (low-carbon electric-furnace steel, annealed), one ring from the outside, one midway, and one from the inside of the cylinder. Accurate measurements were made of the outside and inside diameters of each ring, the distance between diametrically opposite punch marks, and the distance between three sets of punch marks 2 inches apart, tangentially. A cut was then made (by a hand hack saw) midway between the 2-inch tangential marks and at an angle of 90° to the diameter measured. When split thus the outer ring contracted visibly, but the middle and inner rings appeared to undergo no change. Similar rings were cut from casting No. 5 (high-carbon nickel steel, not annealed), measured and cut in the same manner as those from casting No. 4.

TABLE 2.—Survey of Stresses in Concentric Rings from Castings

Part	Casting No. 4			Casting No. 5		
	Before splitting	After splitting	Difference	Before splitting	After splitting	Difference
Outside ring:	Inches	Inches	Inch	Inches	Inches	Inch
Outside diameter	8.080	8.060	0.020 contraction	10.765	10.750	0.015 contraction
Inside diameter	7.280	7.265	.015 contraction	10.205	10.185	.020 contraction
Between punch marks (diameter)	7.650	7.625	.025 contraction	10.515	10.505	.010 contraction
Between tangential punch marks: outer	2.030	1.990	.040 contraction	2.015	1.985	.030 contraction
Between tangential punch marks: middle	2.000	1.960	.040 contraction	1.985	1.950	.030 contraction
Between tangential punch marks: inner	1.920	1.890	.030 contraction	1.990	1.950	.040 contraction
Middle ring:						
Outside diameter	6.840	6.830	.010 contraction	9.530	9.565	.035 expansion
Inside diameter	6.560	6.550	.000	9.080	9.095	.015 expansion
Between punch marks (diameter)	6.920	6.920	.000	9.345	9.355	.010 expansion
Between tangential punch marks: outer	1.985	1.985	.000	2.000	2.015	.015 expansion
Between tangential punch marks: middle	1.955	1.960	.005 expansion	2.005	2.020	.015 expansion
Between tangential punch marks: inner	1.900	1.900	.000	2.000	2.015	.015 expansion
Inside ring:						
Outside diameter	6.790	6.780	.010 contraction	8.515	8.560	.045 expansion
Inside diameter	5.980	5.975	.005 contraction	8.360	8.405	.045 expansion
Between punch marks (diameter)	6.425	6.445	.020 expansion	8.160	8.175	.015 expansion
Between tangential punch marks: outer	1.970	1.965	.005 contraction	1.985	2.025	.040 expansion
Between tangential punch marks: middle	1.955	1.950	.005 contraction	1.975	2.005	.040 expansion
Between tangential punch marks: inner	1.945	1.945	.000	1.975	2.005	.040 expansion

The measurements taken before and after cutting these concentric rings and the amount of expansion or contraction in each case are given in Table 2, and indicate the nature of the stresses in the outer, middle, and inner zones. A rough computation gives the compression at the outside ring of casting No. 4 as 48 000 pound per square inch, and for No. 5 the compression at the outside ring is 31 000 pounds per square inch, and the tension at the inside ring is 47 000 pounds per square inch. These internal stresses are of the order of the elastic limits of the material, and, as would be expected, the outer zone of the casting is in compression.

4. HEAT TREATMENT AND MECHANICAL PROPERTIES

Turning now to the question of the improvement of these castings by heat treatment, in Table 3 are given the details of the treatments to which material from the castings was submitted, and in Table 4 are grouped the mechanical properties associated with the respective treatments. The location of specimens is shown in Figs. 1 and 2. It should be recalled that castings Nos. 1, 3, 4, and 7 were annealed as a whole before any tests were made on them, while castings Nos. 5 and 6 were examined first in the condition as cast. The test pieces from 3*A* and 3*I* were from pieces of the same casting 60 inches apart. An examination of Table 4 shows that most samples show good tensile strength for their composition and treatment and also that there is no marked difference in values for longitudinal and transverse specimens. The values for the Izod shock test are somewhat erratic. The advantages of heat treatment, notably of the double quench and draw, in improving both the resistance to shock and especially the ductility as measured by reduction of area, are strikingly manifest. Certain of these treated steel castings would appear to compare very favorably in their properties with those of forged material of the same compositions. For example, the ordinance requirements for gun forgings are: Elastic limit, 65 000 pounds per square inch; tensile strength, 95 000 pounds per square inch; elongation, longitudinal 22 per cent, transverse 18 per cent; contraction of area, longitudinal 35 per cent, transverse 30 per cent. These are probably more than met by casting No. 6*QQ* (C = 0.33, Ni = 2.75 per cent) and almost met by others such as 3*AQ* and 3*IQ* (C = 0.32, Ni = 2.70 per cent). Again the properties of the casting

7 (C = 0.23 per cent) are high for steel castings of that composition, and in the treated condition are superior to many of the results on hot-rolled 0.20 to 0.25 per cent carbon steels and indeed are comparable to those of cold-rolled steels of this grade.

TABLE 3.—Heat Treatment

Designation of sample	Condition received	Heat treatment given	Temperatures and time of heating: Heated slowly in air at the following temperatures	Then cooled. Held the following number of hours	Temperature when quenched: First	Temperature when quenched: Second	Quenching medium	Drawn 30 minutes at	Cooled in
			°C		°C	°C		°C	
1N	Annealed	Normalized	800–805	2	.....	.....	.....	.....	Air
1Q	do	Quenched and drawn.	800–805	2	795–800	.....	Water	600	Do.
[a]3N	do	Normalized	830–900	3	.....	.....	.....	.....	Do.
3Q	do	Quenched and drawn.	830–910	3	845–850	.....	Oil	640–650	Do.
4N	do	Normalized	910–925	2	.....	.....	.....	.....	Do.
4Q	do	Quenched and drawn.	910–925	2	915–925	.....	Water	450–460	Do.
4QQ	do	Double quenched	910–925	2	915–925	885	do	440–450	Do.
5N	As cast	Normalized	750–760	1½	.....	.....	.....	.....	Do.
5Q	do	Quenched and drawn.	750–760	1½	780	.....	Oil	700–705	Do.
6N	do	Normalized	850–860	2	.....	.....	.....	.....	Do.
6Q	do	Quenched and drawn.	850–860	2	825–830	.....	Water	675–680	Do.
6QQ	do	Double quenched	850–860	2	825–830	790–795	do	680–690	Do.
7Q	Annealed	Quenched and drawn.	875–880	1	875–880	.....	do	510–515	Do.
7QQ	do	do	880–885	1	880–885	845	do	535	Do.

[a] Specimens from A and I sections of No. 3, same treatment.

TABLE 4.—Mechanical Properties

C and Ni	No.	Yield point (Divided by 1000)		Ultimate strength (Divided by 1000)		Elongation in 2 in		Reduction of area		Izod impact		Microstructure	
		L	T	L	T	L	T	L	T	L	T		
		Lb in.²	Lb in.²	Lb. in.²	Lb. in.²					Ft. lb in.²	Ft. lb. in.²		
C, 0.44–0.50; Ni, 2.32–2.39	IX	54.3	.....	91.2		10 5		12.1		48.8		Fairly fine with coarse ferrite boundaries, few slaggy areas	Small blowholes
	IY	53.5	.....	50.5	..	9 5		8.9					
	IT	.....	56.5	...	89.5		5.5		7 5		..		
	IN	67.5	....	93.0	...	8.5		12.5		...		Fine uniform	
	IQ	87.5	91.3	89.0	105.0	4.5	6.5	11.5	14 0	85.2	..	Very fine uniform	
C, 0.32–0.34; Ni, 2.66–2.75	3AX	53.8	.....	89.8		7.5		12.1		252.0	187.0	Fir-tree crystals and a few slaggy ferrite areas.	Ingotism
	3AY	48.5	.....	81 5	.	16.5	....	22.0	..	.			
	3AT	.....	50.7	. .	78.2	....	26.0		42 5	.			
	3AN	59.8	.....	95.7		16.0		29.3	. .	.	....	Fine uniform.	
	3AQ	67.5	70.2	98.0	90.3	21.5	20.5	47.4	42.0	309.0	704 0	Sorbitic, ferrite ghosts	
C, 0.30–0.35; Ni, 2.66–2.76	31X	52.5	.....	88.7	. . .	14.8		19.3	.	...	.	Finer than 3A.	Very few slaggy areas
	31Y	48.4	. . .	81.8	..	27.0		41.8	.	. .	...	Fairly uniform	
	31N	60.5	.. ..	89.1		9.5		18.5	..	. . ...	..	Fine, uniform.	
	31Q	64.5	.....	95.5	....	21.0		29 9	. .	.. ...	..	Do.	
C, 0.16–0.21....	4	40.0	.....	61.5	. ...	16.5		10.4	....	34.5	48.4	Coarse with thick ferrite veins	
	4N	47.5	.....	65.0	...	11 5		18.9	..	.....	.....		
	4Q	60.0	. . .	79.2	. .	13.0		20.3	...	247.0	304.0	Fine, trace of veins	
	4QQ	55.0	.....	72.5	. .	12.5		24.5	.	. ...		Fine with streaks	
C, 0.63–0.72; Ni, 2.90–2.94.	5	48.0	65.7	85.0	84.7	2.5	1.0	1.2	1.0	16.3	16.1	Coarse pearlite	
	5N	77.5	. . .	129.0	.....	6.5		5.4		.....	.....	Fine slaggy pearlite areas	
	5Q	96.4	92.3	115.0	113.3	16.5	7.5	21.7	10.5	142.0	183.0		
C, 0.31–0.35; Ni, 2.69–2.81.	6X	47.5	. . .	82.5	. . .	17 5		21.5		432.0	.	Fairly fine; ferrite veins, slaggy areas	
	6Y	48.0	.....	81.5	...	17.5		28.5	. . .	. .	.		
	6T	.... .	50.0	. .	83.0	. .	16.5	..	22 0	..		Ingotism	
	6N	60.0	... .	87.5	.	14.5		22.0	....	....	...	Fine; trace ferrite ghosts	
	6Q	83.5	85.0	100.5	104.0	16.5	12.5	26.5	24.5	452.0	456.0	Very fine, pitted with holes	
	6QQ	70.0	.....	92.0		23.0		44.0		.	..	Do.	
C, 0.22–0.25....	7N	38.0	. . . .	64.0	. . .	35.5	.	52.5	.	....	.....	Fine, uniform	
	7Q	69.3	.....	95.4	....	18 3		39.2	...	.....	.....	Very fine, sorbitic, ferrite	
	7QQ	57.8	.....	79 0	...	27.3	. .	47.7	.	.....	.....	Very fine, all sorbitic	

X inner zone. Y outer zone. T tangential. N normalized. Q quenched and drawn. QQ double-quenched and drawn.

5. MACROSTRUCTURE AND MICROSTRUCTURE

It is of interest to note there appeared no flaws or visible defects, other than small blowholes near the inner surface, in the preparation of any of the test pieces for physical or chemical examination. The blowholes noted were always within about one-sixteenth inch from the inner or free surface of the casting. No hard spots were found in any of the castings. Just next the inner surface of all castings there is a layer not over one-sixteenth inch thick which appears to contain nearly all the physical and chemical discontinuities.

Sulphur prints and cupric ammonium chloride etchings were made on transverse sections of each casting. Some of the sections photographed are shown on Figs. 3, 4, and 5, and represent the coarsest grained and most irregular portions of the sections

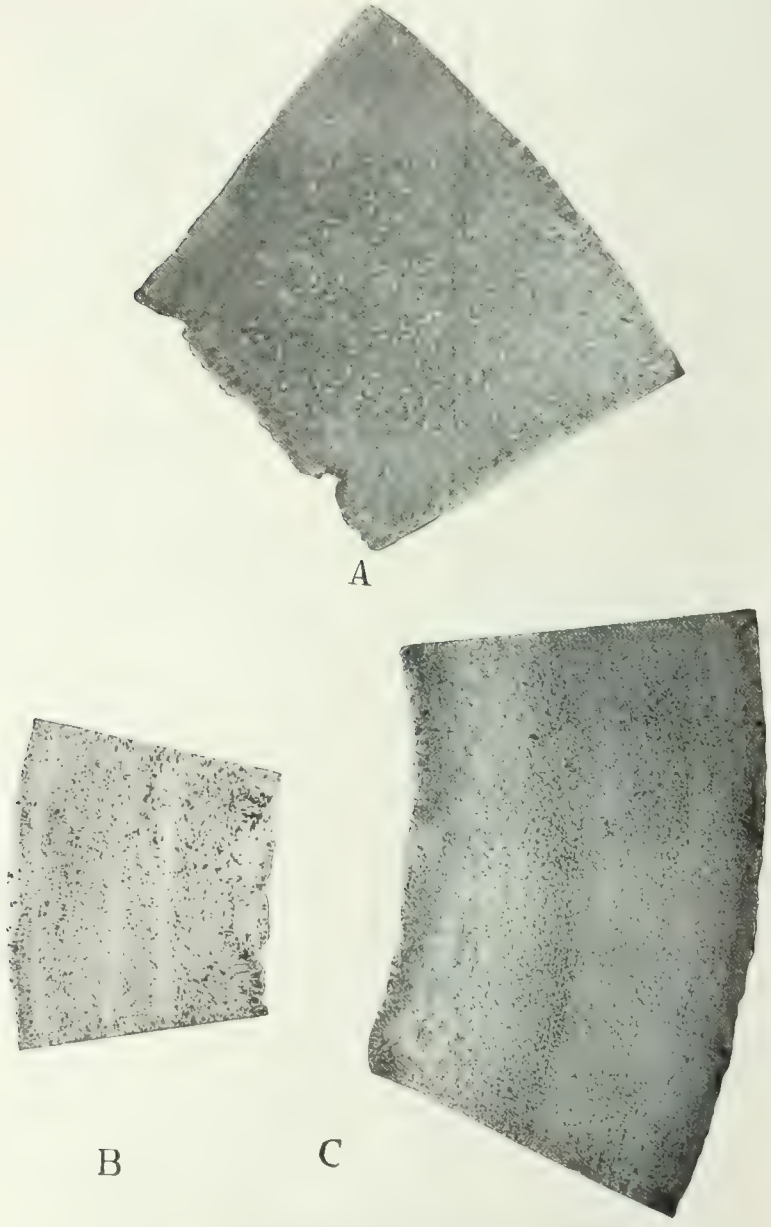


FIG. 3.—*Transverse sections of castings Nos. 1 and 5*

(A) Casting No. 5. Nitric acid etching of a transverse section (×, about 1½). The structure is coarse grained with higher carbon areas in the inner zone

(B) Casting No. 5. Sulphur print of a transverse section. There are small high-sulphur areas in the inner zone and several circumferential bands low in sulphur in the middle and outer zones

(C) Casting No. 1: Cupric ammonium chloride etching of a transverse section. This structure is practically free from fir-tree crystals, but shows circumferential bands or zones apparently highest in carbon in the inner zone and lowest in carbon in the middle zone

FIG. 1.—*Portion of transverse ring from casting No. 1.*

(A) Sulphur print, actual size. There is little variation in the amount of sulphur, slightly more along inner edge than near outer edge. Several small local segregations may be seen.
(B) Cupric ammonium chloride etching. This shows the coarse-grained structure of the steel.

examined. The 72-inch longitudinal section was finished and rubbed down with fine emery paper, and a sulphur print was taken along the entire length; no longitudinal segregation was found.

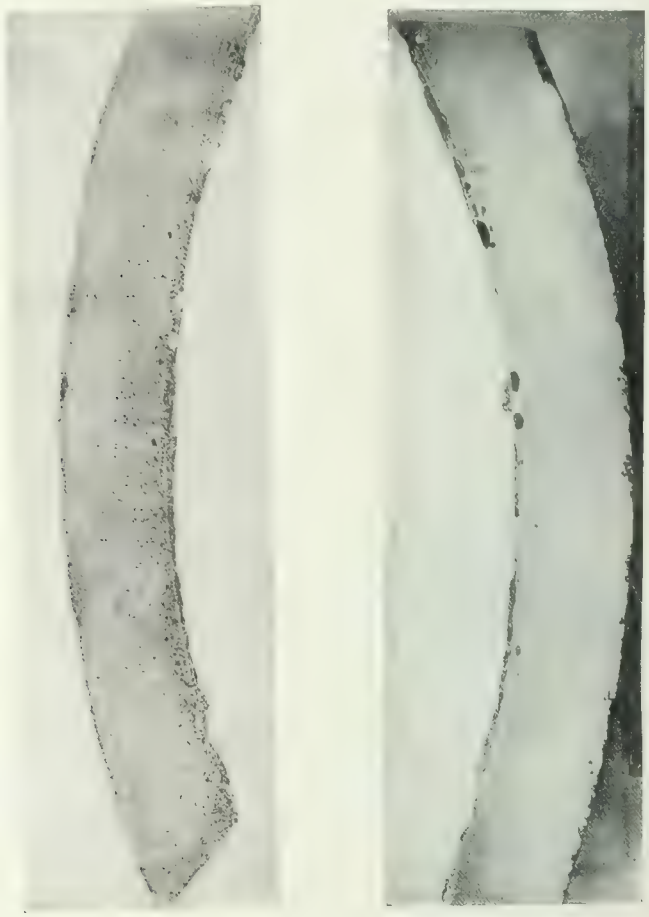


FIG. 5.—*Portion of transverse section of casting No. 7*

(A) Sulphur print, actual size. This shows practically no segregation of sulphur except a few small spots at the extreme inner edge where small surface blowholes occur.

(B) Cupric ammonium chloride etching, actual size. This is a portion of the transverse section (full thickness of cylinder as cast, no machining) and shows the steel to be free from ingotism. There are small surface blowholes, all within $\frac{1}{16}$ inch of the surface. The inner zone (zone 3) is slightly higher in carbon.

At one point a hole or blister, 2 inches long, $\frac{1}{8}$ inch wide, was found $\frac{3}{32}$ inch from inside edge of the casting and a distance of 6 inches from one end. Fig. 3A, a nitric acid etching of the entire section,

revealed nothing that is not shown in the sulphur prints. Any appreciable segregation in the casting occurs transversely from outside to inside, as shown in the etched transverse section.

Specimens for microscopic examination were cut from transverse, radial (longitudinal), and tangential sections of each casting as received, and from transverse sections of the heat-treated bars; that is, after normalizing, quenching, and drawing, and double-quenching and drawing. Photomicrographs taken at a magnification of 50 diameters are shown and described in Figs. 4 to 9*a*. Each casting was found to contain relatively large ferrite areas (locally decarburized areas) or long ferrite veins (outlining very coarse primary crystals).

The most serious of the low-carbon areas were found in castings Nos. 3*A* and 6 (Figs. 7 and 8). These areas are not completely eliminated by normalizing, and persist even after quenching and drawing, but have been almost completely eliminated by the double-quench and draw treatment. The normalizing treatment, for the most part, greatly refines the structures, while quenching and drawing produces a sorbitic structure with partially diffused ferrite. Structures after the double-quench and draw compare favorably with those of large heat-treated steel forgings.

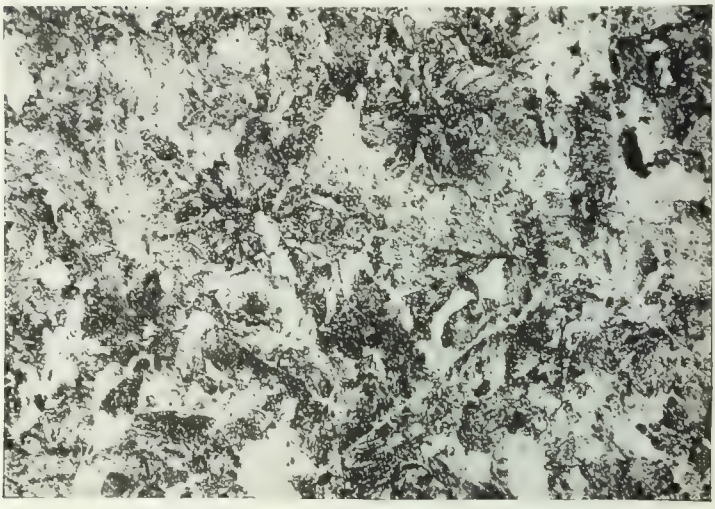
IV. SUMMARY

There were examined a miscellaneous lot of five castings in the form of cylinders of wall thickness 8½ inches to ½ inch, made by the Millspaugh centrifugal process. Their composition ranged from two low-carbon steels, C = 0.17 and 0.23 per cent, to three nickel steels ranging in carbon from C = 0.33, 0.46, to 0.66 per cent, and in nickel from Ni = 2.69, 2.35, to 2.92 per cent.

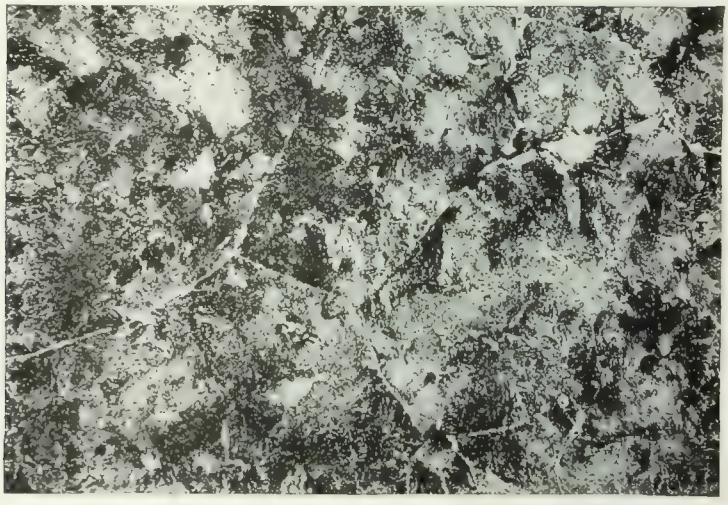
Segregation of the elements carbon, phosphorus, sulphur, nickel, and copper appears to exist to a slight extent, but only radially, and is most marked next the inner surface in a narrow zone of about one-sixteenth inch depth. Manganese and silicon do not segregate in this type of casting. The greatest carbon segregation was 0.09 per cent in nickel steel (C = 0.66, Ni = 2.92 per cent) and the least was 0.02 per cent in another nickel steel (C = 0.33, Ni = 2.69 per cent).

The *hardness* (Brinell and scleroscope) practically follows the segregation.

The only evidence of *unsoundness* was the presence of small blowholes in the inner zone, usually within one-sixteenth inch of surface.



A

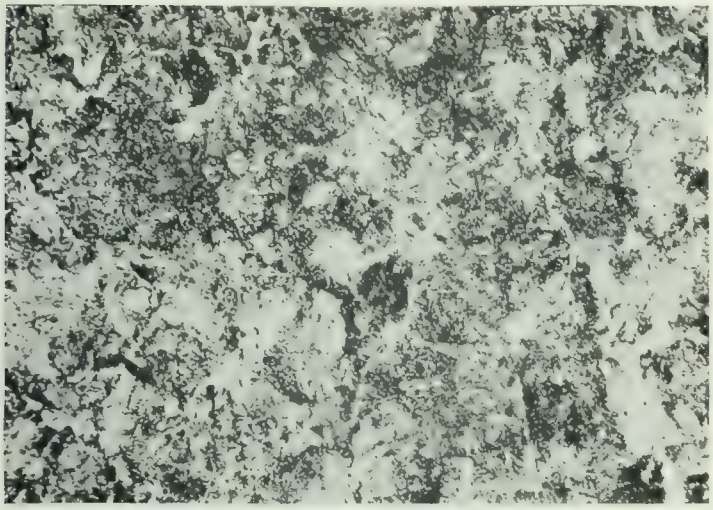


B

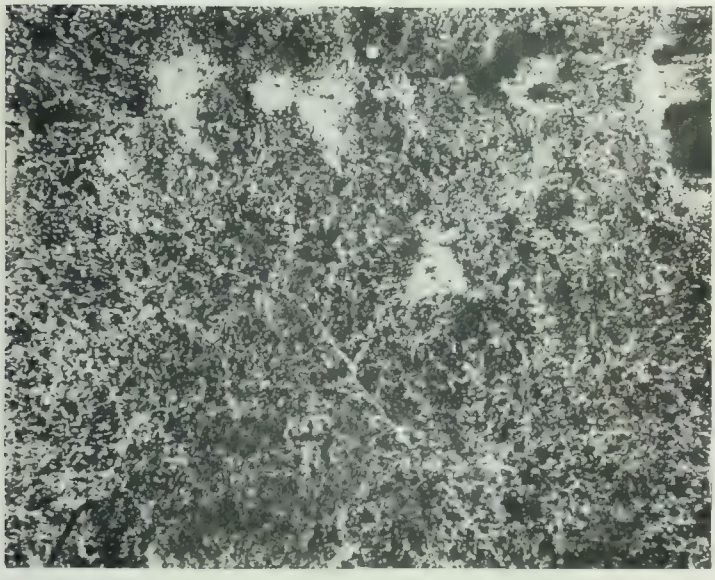
FIG. 6.—*Castina No. 5. Etched with picric acid.* × 50

(A) Transverse section. This steel is very coarse grained and apparently has not been annealed. The average structure consists of pearlite with a thin, polyhedral network of ferrite and is fairly uniform except the zone (about ¼ inch deep) adjacent to the bore, where there are large areas eutectoid in structure about 10 points higher in carbon than the average structure.

(B) Radial, longitudinal section. This is similar to the transverse section.



C



D

FIG. 6a.—*Casting No. 5 as cast and normalized. Etched with picric acid.*

× 50

(C) Tangential section as cast. This structure is slightly finer than that of the transverse and radial sections (see Fig. 6)

(D) Normalized. The grain has been greatly refined by this treatment, but traces of ferrite areas containing slag inclusions still remain. The ferrite network has not been entirely eliminated. A few small blowholes may be seen

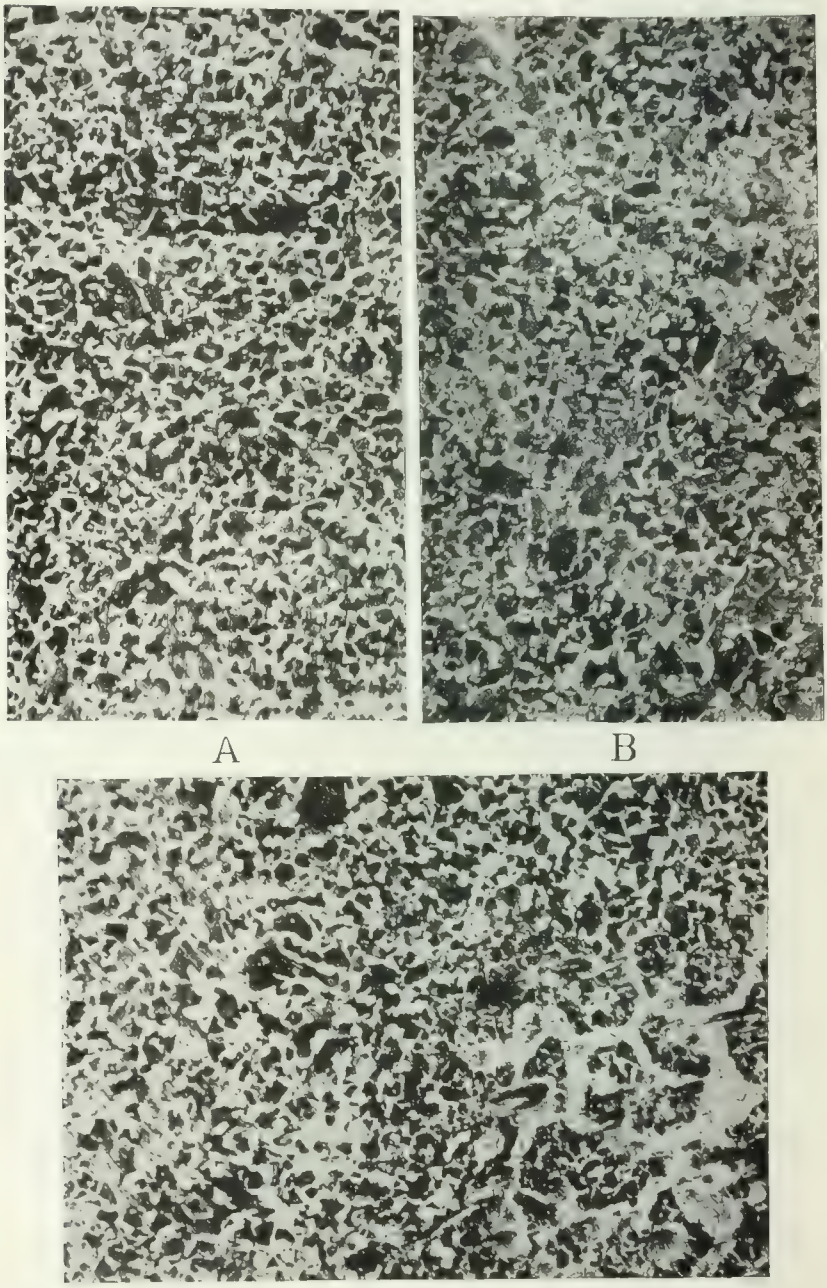


FIG. 7.—*Casting No. 6 as cast. Etched with picric acid.* × 50

(A) Transverse section. This casting, although not annealed, has a fine-grained structure. Traces of fir-tree crystals are found near the inner edges, and coarse primary crystals are outlined by a thin ferrite network, but no slaggy ferrite areas appear in this section

(B) Radial section. This shows more pronounced fir-tree crystals than the transverse section

(C) Tangential section. A cluster of slaggy low-carbon areas appears in this section. Traces of fir-tree crystals were found also

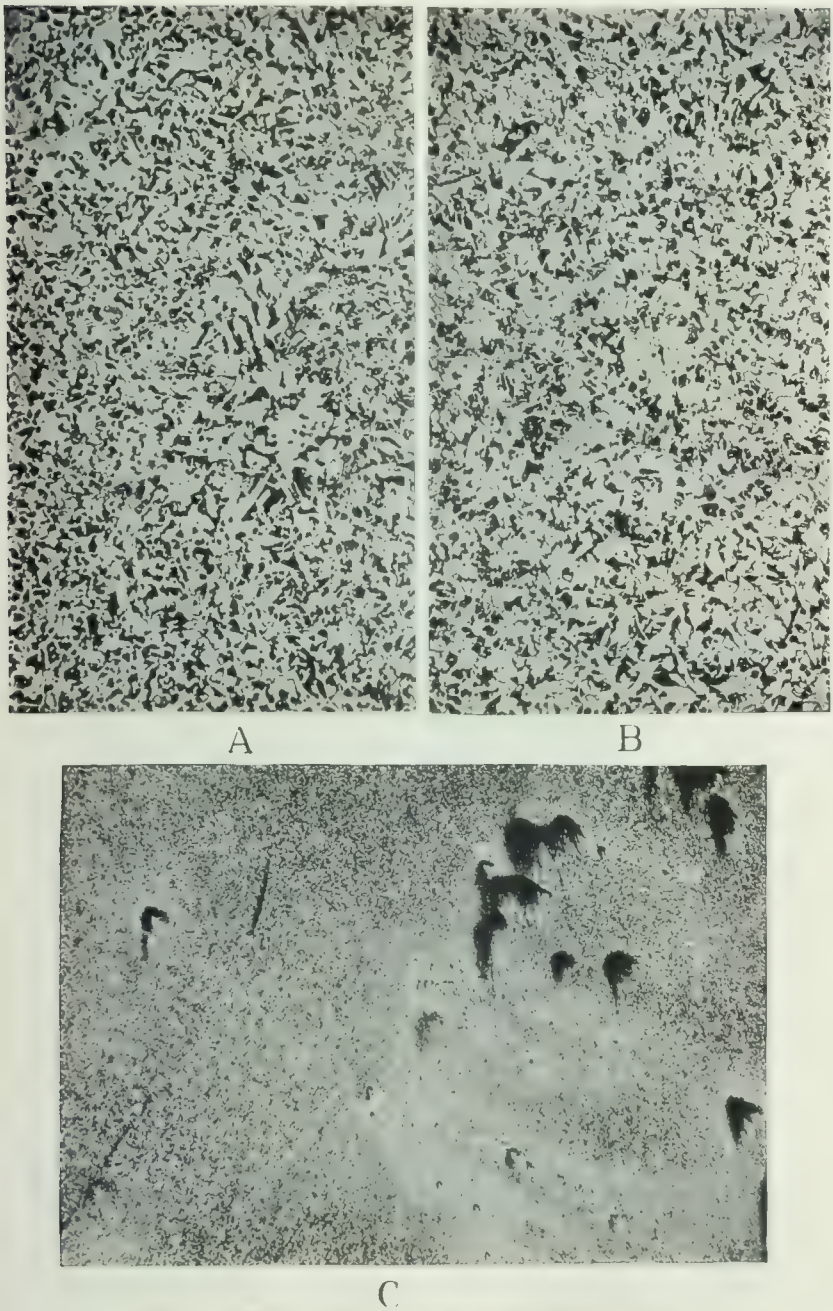


FIG. 8.—*Casting No. 6 etched with picric acid.* × 50

(A) Normalized. A fairly uniform, fine-grained ferrite-pearlite structure has been produced. No ferrite veins were found

(B) Water quenched and drawn. This structure is sorbitic with a fine network of undiffused ferrite. Slight traces of ferrite veins, partially broken up, were found

(C) Double-quenched and drawn. Although the structure has been greatly refined and the ferrite well diffused, shadowy traces of ferrite veins remain. An area pitted with tiny holes is here shown

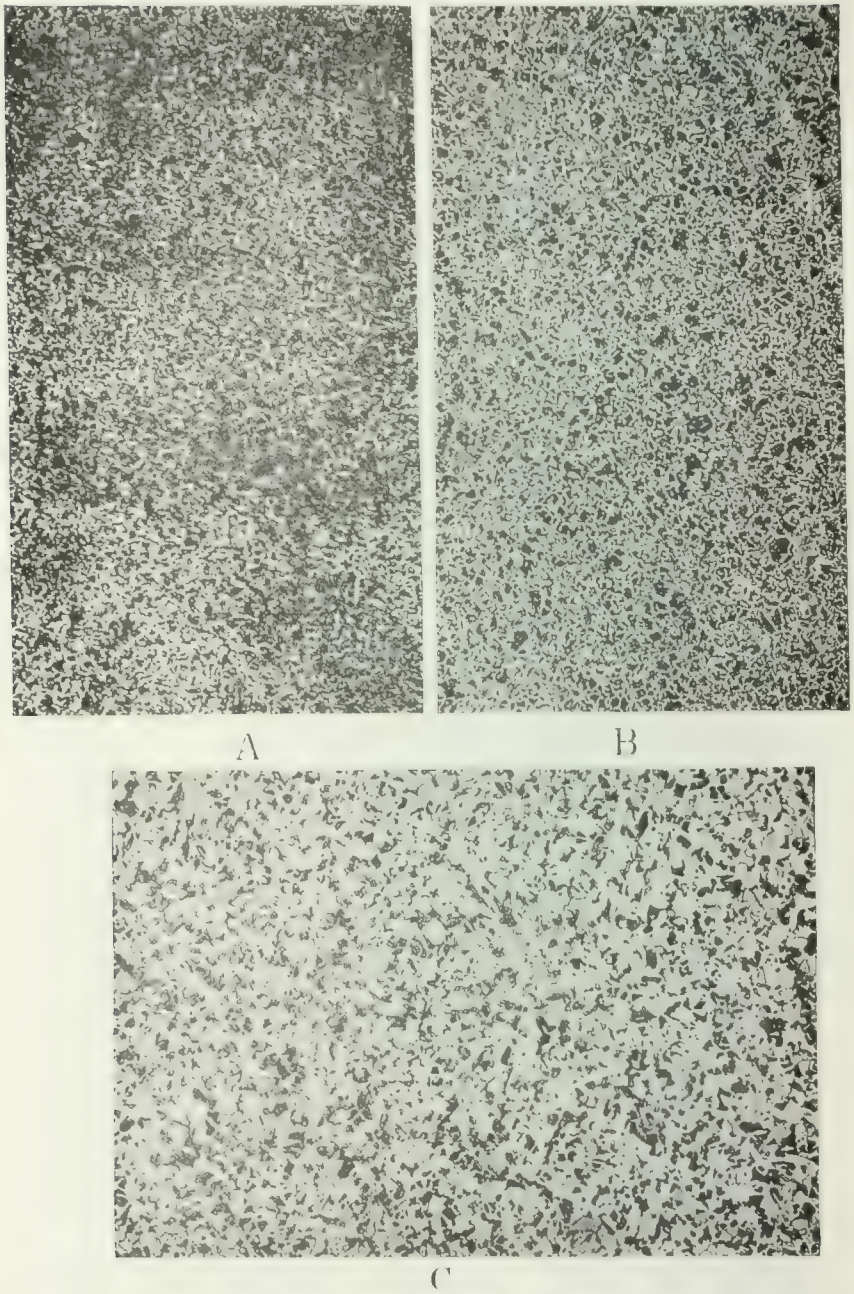


FIG. 9.—*Casting No. 7 (basic open-hearth steel etched with picric acid.* × 50

(A) Transverse section, as received. There is a fine grained uniform ferrite pearlite structure free from blowholes, segregation, or slag. The steel has been well annealed

(B) Radial longitudinal section as received. This structure is similar to that of the transverse section

(C) Tangential section as received. The structure is fine grained but has a few areas containing needle-like ferrite particles, indicating incomplete annealing

D

E

FIG. 9a. *Casting No. 7. Transverse section etched with picric acid* ×50

(D) After single heat treatment. The steel has a very fine sorbitic structure with a ferrite network, and a few ferrite areas not fully broken up

(E) After double heat treatment. The structure is fine and sorbitic. The network has been broken up, but many ferrite areas still remain

The *density* across a section is practically constant; that is, to 0.004 in 7.836.

The *mechanical properties* show, in general, somewhat greater strength, elastic limit, and resistance to shock, but less ductility, in the tangential than in the longitudinal direction.

The *internal stresses* developed in the castings by this centrifugal process of manufacture, as determined from measurements on three rings from each of two castings, show values of the order of the elastic limit with the outer zones in compression.

The effect of *heat treatment* in improving the physical properties of the castings is very marked. The results suggest that the properties of such castings suitably treated may rival those of forgings of the same chemical composition.

The *microstructure* of at least some of these castings is better than that of ordinary castings; certain ones show pronounced ingotisms (dendritic structure). The nickel steels contain more slag inclusions than is usual in ordnance steel, showing that this centrifugal process may not clear up a basic steel. The ingotism and coarse grained structures of these centrifugal castings can, in general, be removed only by prolonged and repeated heat treatments; that is, normalizing followed by double-quench and draw.

The above account is limited to a description of the tests, and no references are given as to the details of the manufacturing operations, since we had no first-hand information concerning them.

Acknowledgments should be made to the members of the staff of the divisions of chemistry, structural materials, and metallurgy who assisted in these tests and in particular to G. N. Nauss, who followed them through in detail.

WASHINGTON, February 11, 1921.

DEPARTMENT OF COMMERCE

TECHNOLOGIC PAPERS

OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 193

DESIGN OF ATMOSPHERIC GAS BURNERS

BY

WALTER M. BERRY, Gas Engineer
I. V. BRUMBAUGH, Associate Gas Engineer
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Bureau of Standards

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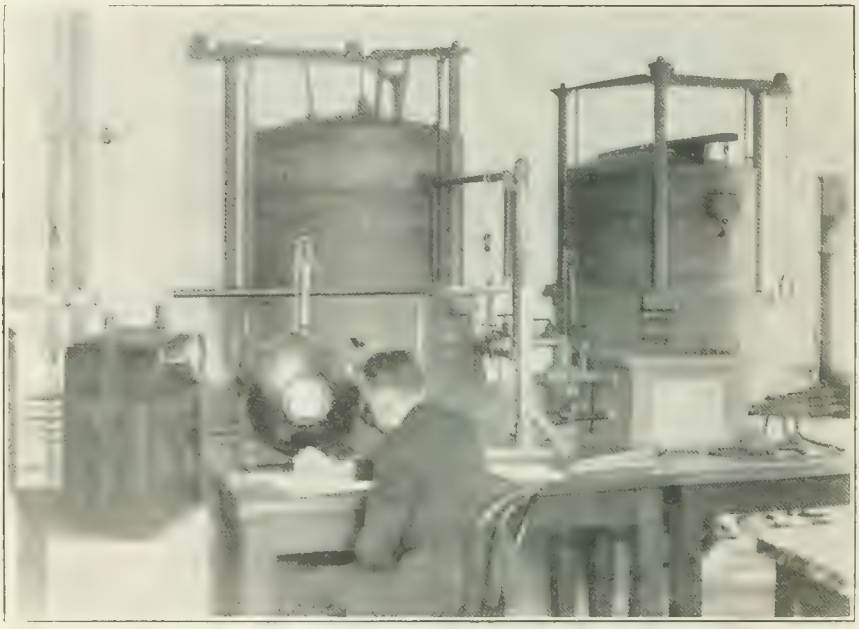

FIG. 1.—*Appliance laboratory, Bureau of Standards, showing apparatus used to test burners*

DESIGN OF ATMOSPHERIC GAS BURNERS

By Walter M. Berry, I. V. Brumbaugh, G. F. Moulton, and G. B. Shawn

ABSTRACT

The first part of an extensive investigation of the design of gas burners is presented in this paper. With the arrangement of apparatus and method of testing that has been developed one can measure quickly and accurately the volume of air injected into any burner under any condition of operation, as well as determine the limits of operation with any quality of gas. Such information is essential in order to enable one to design burners for any predetermined condition of operation.

In order to understand the various factors entering into the design of burners it was found necessary to study the theory of flow of gas through different types of orifices, the principles governing the rate of injection of air into the burner, the design of the injecting tube, the rate of consumption of burners of different port areas, and the effect of adjustment of the air shutter.

The rate of discharge of gas orifices of different types was found to vary greatly with a variation of the angle of approach and with the length of channel or tube of the orifice.

The advantages of having an injector of good design to secure the injection of a larger volume of primary air have been illustrated by tables and a large number of curves. The general design of an injecting tube that produced the greatest injection of primary air is shown.

It has been found that for any given burner the ratio between the momentum of the gas stream and the momentum of the stream of the air-gas mixture entering the burner is always a constant, and this relation enables one to calculate readily the effect on the volume of air entrained when the gas pressure, gas rate, or specific gravity of the gas is changed.

It is very important that there should be a correct relation between the area of the throat of the injecting tube and the area of the burner ports if the energy of the gas is to be efficiently utilized to inject air into a burner. The results of the tests show that the area of the injector throat should be about 43 per cent of the area of the burner ports.

In pipe burners the rate of consumption per square inch of port area increases until the port area is about equal to the cross-sectional area of the pipe. Tables have been tabulated to show the rate of consumption of burners with and without improved injecting tubes.

If the openings in the air shutter are too small there is an appreciable loss of air injected into the burner. The velocity through the air opening should not exceed 4 or 5 feet per second.

The results of tests to show the efficiency of operation of various types of burners with different qualities of gas will be the subject of future reports.

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I. INTRODUCTION

1. PURPOSE OF INVESTIGATION

The large increase in cost of gas-making materials and the difficulty in securing materials of satisfactory quality have resulted in frequent increases in the price of gas to the consumer. This has led to attempts to secure higher efficiencies with the existing appliances, and to devise new and improved methods of utilization.

In many localities the shortage of raw materials has compelled a decrease in standards of quality, and in other localities they have lowered the standards instead of increasing the price. The problem of determining the best standard for any locality would be made materially easier if definite information on the relative utilization efficiencies of different qualities of gas were available. This question can be solved only by a careful study of the performance of appliances and by determining the adjustments which will give the highest efficiency with each quality of gas.

The nation-wide movement for the conservation of natural gas has already brought out rather clearly the fact that a great saving of natural gas can be accomplished by improvement in the design and by readjustment of natural-gas appliances. Many of the natural-gas companies are also finding it necessary to supplement their supply with artificial gas, and, since the characteristics of artificial and natural gas are so different, some investigation of the appliances has been necessary to determine within what limits it is possible to vary the proportions of the two gases.

The question of increasing the efficiency of utilization of gas has been considered so important that a great deal of effort of the Industrial Fuel Sales Committee of the American Gas Association has been directed toward research work on this problem. As a part of this program the gas engineering section of the bureau of standards was requested to make a study of atmospheric burner design and operation. The authors desire to acknowledge the assistance and encouragement which has been received from the members of that committee. Credit is also due J. H. Eiseman for efficient services in this investigation.

A brief report of the first results of this work was read before the commercial section at the 1919 convention of the American Gas Association.

2. SCOPE OF TESTS AND OF THIS PAPER

This paper is based on several thousand observations in which the effects of many of the large number of variables affecting burner operation were studied under a wide range of operating conditions. After a description of the principles of operation of atmospheric burners, the results of investigations on the effect of design of orifice and injecting tube on rate of consumption of burners are given, as well as a discussion of the general principles governing air injection. The questions of burner design, involving the relations between port area and rate of consumption of burners, the limit of velocity of efflux from the ports, and other factors affecting the operation of burners are also considered. Some of the other phases of this problem, such as the completeness of combustion, flame characteristics with different gases, efficiency of operation of burners of various designs and using gases of different quality and with different proportions of primary air, which are not covered in this paper, will be discussed in later reports.

The subject of industrial atmospheric burner design has been covered in a general way in the lesson papers of the educational course edited by the American Gas Association, but these articles do not attempt to show numerically the effects of changes in the gas rate, gas pressure, or the specific gravity of the gas. The advantage of a correctly designed injecting tube as a means of improving the combustion and securing greater rate of consumption has evidently not been fully appreciated, although specially designed injecting tubes have been used to some extent to advantage.

No extensive presentation of theory has been attempted, the aim being rather to show by means of tables and curves representing experimental data the effect of the various factors on the operation of burners. The tables and curves, unless otherwise indicated, will be based on actual results of tests, and can be duplicated by anyone who may care to continue any part of the investigation.

Some of the following discussion may seem elementary, but our experience has shown that the problem is difficult because of the large number of variables involved and the difficulty of keeping in mind the proper relation between the several factors. For that reason an attempt has been made to show clearly by means of illustrations and solutions of simple problems the effect of each variable.

It is hoped that the information in this paper will be of practical assistance to the manufacturers of gas appliances, as well as to the industrial gas-appliance engineers who are frequently required to design and make their own burners for many special purposes.

II. DESCRIPTION OF APPARATUS AND METHODS OF TEST

A photograph of the apparatus used at this Bureau where these tests were made is shown in Fig. 1. The arrangement of the apparatus is shown in Fig. 2.

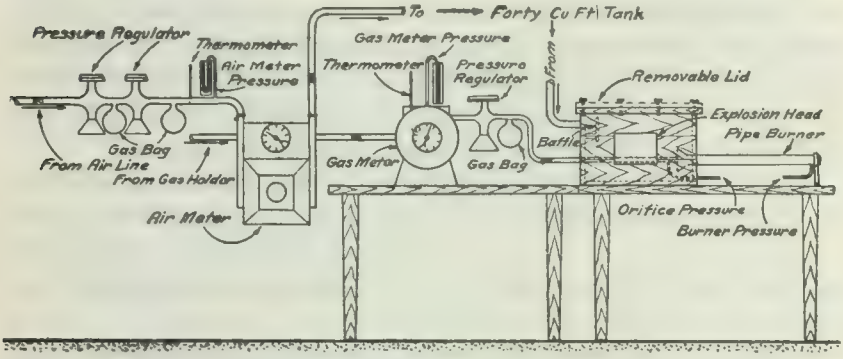


FIG. 2.—*Arrangement of apparatus used for burner tests*

1. METERS AND REGULATORS

In order to obtain higher pressures at the orifice than were available from the gas supply line, it was found necessary to pump the gas from the line into a weighted 40 cubic foot gas holder. The gas was measured with a carefully calibrated wet meter. The temperature and pressure of the gas were read at the time of each meter reading. At the outlet of the meter there were placed in series a dry regulator, and a special gas bag which served as an excellent antifluctuator. This combination gave such uniform pressure at the orifice that the slight kicks from the meter were hardly perceptible in the burner pressure, except at the higher gas rates.

The air supply was taken from a regular laboratory supply line. The pressure was reduced by two dry regulators in series. A gas bag connected at the outlet of each regulator eliminated the remaining fluctuation caused by the compressor. A 30-light dry meter was used for measuring the air. To eliminate the fluctuations in pressure caused by this meter the air was allowed to

flow through a 40 cubic foot tank. The temperature and pressure were recorded at each observation.

At the beginning of these tests considerable difficulty was experienced from the fluctuation in pressure in both the gas and air lines, as well as the kicks from the meters. By the use of these regulators and gas bags the air and gas entered the burner at such uniform pressure that the greatest fluctuations of pressure in the burner were usually less than 0.001 inch of water pressure.

2. METHOD USED FOR MEASURING AIR-GAS RATIO

It was realized soon after the experiments were started that the determination of the volume of primary air injected, by determining the percentage of air in the mixture, would be burdensome. This method was attempted at first but found to be very slow, and the results did not agree even though extreme care was used in drawing the samples of the mixture from the burner.

After considerable experimentation a method was devised for determining the air-gas ratio by which it was possible to gather a large quantity of data with speed and precision. Since this method might be very useful in other similar investigations, it is described in detail.

It was thought that by inclosing the inlet of the burner in an air-tight box and metering the air into the box at a rate which would keep the pressure inside at exactly atmospheric pressure, the volume of air entering the burner could be determined directly from the meter reading. This method was described by Mansfield in London Journal of Gas Lighting, July 13, 1909. It was found, however, that minute variations in pressure in the box which could not be determined on the most sensitive gage that was available gave very wide variations in the ratio of the air to gas mixture entering the burner.

The plan tried next, which has been used very successfully, was to leave the top of the box open and operate the burner in a normal manner, observing carefully the pressure in the burner as indicated on a very sensitive slope U gage. The lid was then placed on the box and the air passed through a meter into the box at a rate which exactly duplicated the previous conditions of pressure within the burner. It is evident that if the gas rate in each case was the same, and the pressure in the burner was exactly duplicated, the volume of air injected into the burner in each case was the same. The ratio of the volume of air to the volume of

gas that entered the burner was thus readily calculated from the readings of the two meters.

Xylene was used in the slope U-gage. The gage was calibrated against a water gage and was found to give excellent results.

When it was desired to note the flame characteristics, the heights of the blue inner cone and of the flame were also measured.

In some tests made with an open pipe burner, where the static pressure was low and the velocity pressure was high, a very small bent tube was inserted into the center of the pipe burner, with the open end of the tube against the stream. The velocity pressure in the burner was thus secured. Since, in practically all of this work, the absolute pressure values were not essential, it made little difference how or where the pressures were measured so long as satisfactory and comparable relative readings were obtained. It was gratifying to find that with our apparatus and method it proved possible to make the various measurements with considerable precision and reproducibility, as indicated by the tables and curves in this paper.

To eliminate any hazard in case there should be an explosion in the box caused by a combustible mixture, a piece of oiled paper was pasted over a large opening in the side of the box and made a very satisfactory explosion head. It was demonstrated on two occasions during the tests that this was a very wise precaution.

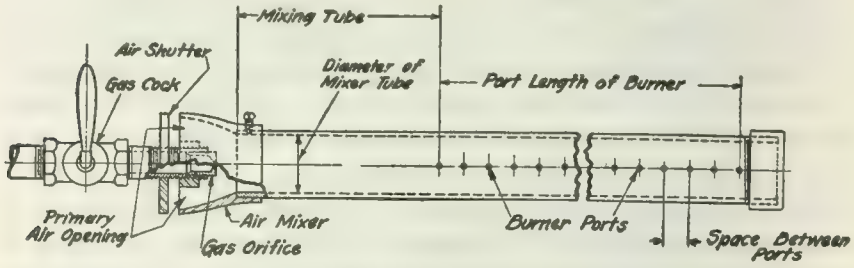
III. THE OPERATION OF AN ATMOSPHERIC BURNER

1. DESCRIPTION OF BURNER

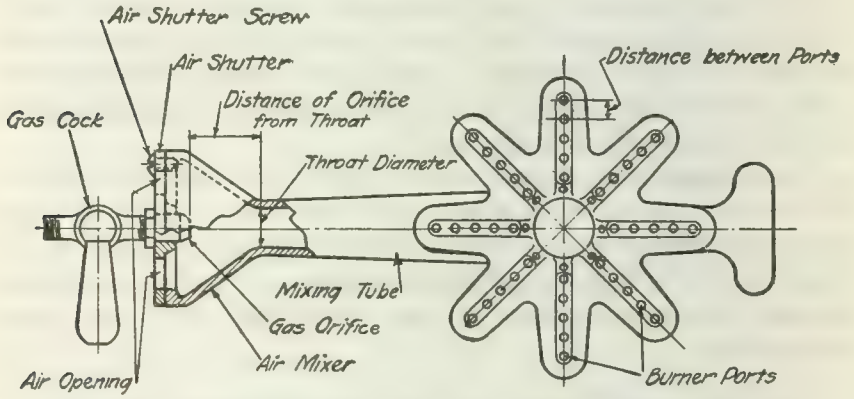
In Fig. 3 is shown a simple type of atmospheric pipe burner used in these tests, and also a sketch of the ordinary domestic range burner, to which much of the following discussion is equally applicable. The different parts of the burners are clearly designated according to the nomenclature commonly used by gas men.

The gas, after passing through the controlling cock, issues with a high velocity through the orifice (about 160 feet per second, with 4-inch pressure and gas of 0.65 specific gravity) and the momentum of the gas stream causes air to be injected into the burner. The velocity pressure of the mixture is converted into static pressure sufficient to force the mixture through the burner ports.

The mixture of gas and air passes through the constriction or injecting throat into the mixing tube, where the gas and air are fairly well mixed before arriving at the first ports in the burner proper, or burner head, as it is called in burners of the domestic or ring burner type.



(*a*) Atmospheric pipe burner—industrial type



(*b*) Atmospheric star burner—domestic type

FIG. 3.—*Top view of atmospheric burners of industrial and domestic types, showing nomenclature commonly used*

2. FACTORS AFFECTING AIR ENTRAINMENT

The total volume of air entrained by a stream of gas depends upon the following factors:

1. Gas rate.
2. Gas pressure.
3. Specific gravity of gas.
4. Design of orifice—determining the friction loss.
5. Position of orifice.
6. Area of air opening in shutter.
7. Design of injecting tube.
8. Dimensions and shape of burner head.
9. Total area of flame ports.
10. Temperature of the burner head.

The effect of gas pressure, rate of flow, and specific gravity of the gas on air entrainment will be discussed under the heading "Principles governing air injection"; that of the other factors will appear in later sections under the appropriate headings.

IV. THE BURNER ORIFICE

1. LAWS OF FLOW OF GAS THROUGH ORIFICES

Many investigators, in their calculations of the velocity of flow of gas through an orifice, start with the assumption that the flow is adiabatic. It is undoubedly true for gases that the flow follows that condition, but for small pressures, such as are used in an ordinary city supply, the difference in absolute pressure is very small, and, therefore, the difference in specific volume due to compressibility is very small, and the hydraulic formula can be applied with an error less than 1 per cent.[1] In this case the "theoretical velocity" of flow in feet per second through an orifice is

$$V = \sqrt{2gh},$$

in which h = the "head" or height in feet of a column of gas (of the same density as the gas at the orifice) required to produce the pressure at the level of the orifice, and g = the acceleration of gravity—32.2 feet per second, per second. The actual velocity at

---

[1] See B. S. Sci. Paper No. 359, Efflux of gases through small orifices, by E. Buckingham and J. D. Edwards for the development of the following adiabatic formula of velocity in feet per second.

$$V^2 = 2g\frac{P_1}{W}\cdot\frac{1-r^x}{x}, \text{ where}$$

P_1 = initial absolute pressure of the gas in pounds per square foot;
W = initial density of the gas in pounds per cubic foot;
$x = \frac{k-1}{k}$, where $k = \frac{C_p}{C_v}$ the ratio of specific heats and is very approximately equal to 1.40;
$r = \frac{P}{P_1}$, where P = the back pressure, which in this case is merely the outside barometric pressure in pounds per square foot.

Let $P = P_1 - p$, where p is the difference between the initial and barometric pressure in pounds per square foot. Then for small values of p such as would correspond to city supply pressures, we have very approximately

$$r^x = \left(\frac{P}{P_1}\right)^x = \left(\frac{P_1 - p}{P_1}\right)^x = \left(1 - \frac{p}{P_1}\right)^x = 1 - x\frac{p}{P_1},$$

in which case

$$V^2 = 2g\frac{P_1}{W}\cdot\frac{1-\left(1-x\frac{p}{P_1}\right)}{x} = 2g\frac{P_1}{W}\cdot\frac{x\frac{p}{P_1}}{x}$$

$= 2g\frac{p}{W}$, where $p = Wh$, $\therefore h = \frac{p}{W}$ and the equation

$V = \sqrt{2g\frac{p}{W}}$ is identical with the hydraulic equation

$V = \sqrt{2gh}$.

The error that will result from using $r^x = 1 - x\frac{p}{P_1}$ is less than 0.1 per cent for a pressure of 4 inches of water and is about 0.4 per cent for a pressure of 8.0 inches of water.

the orifice is always appreciably less than this theoretical velocity, and consequently the quantity of flow is less than the quantity calculated from the theoretical velocity and area of orifice. The ratio of the actual flow to the theoretical is known as the discharge coefficient. The quantity of flow per second is, therefore, equal to the product of the theoretical velocity, the area of the orifice, and the discharge coefficient, which latter takes into account the contraction of the vein or flowing stream, the friction of the orifice, etc.

With a stream of water, using a sharp-edged orifice, such as has been designated as No. 3, Fig. 6, in this paper, it is reported in Marks' Mechanical Engineers' Handbook that the minimum area of the cross section of the jet, which is at a distance out from the orifice about one-half its diameter, is about 0.62 of the area of the orifice. It is stated that the average velocity at this point is about 0.98 to 0.99 of $\sqrt{2gh}$. The discharge coefficient, which is the product of the contraction coefficient and the velocity coefficient, for the above condition is 0.61. With an orifice of this type and using the above formula, our tests have shown the discharge coefficient with a gas stream to be about 0.605, which is identical for the value given for liquids.

A simple equation by which to compute the theoretical velocity of gas from an orifice can be developed as follows:

Let H = orifice pressure of gas in inches of water;

d = specific gravity of gas (air = 1.0); then

$$h = \frac{H}{12}\,\frac{820}{d} = 68.33\,\frac{H}{d} \text{ feet, where}$$

$$820 = \frac{\text{density of water at 60° F.}}{\text{density of air at 30 inches Hg and 60° F}}$$

Therefore, the theoretical velocity is

$$V = \sqrt{2gh} = \sqrt{64.4 \times 68.33\,\frac{H}{d}},$$

$$= 66.34\sqrt{\frac{H}{d}} \text{ feet per second.}$$

(*a*) EFFECT OF PRESSURE ON RATE OF FLOW OF GAS

From this simple equation it is at once apparent that the theoretical velocity, therefore, the rate of flow of gas of a given specific gravity issuing from an orifice, varies as the square root of the pressure. Thus, if it is assumed that 1 cubic foot of gas per unit of time flows through an orifice at a pressure of 1 inch the volume

of gas of the same specific gravity that would flow through the same orifice per unit of time at different pressures would be with 2 inches pressure 1.415 cubic feet, with 4 inches pressure 2.000 cubic feet and with 5 inches pressure 2.235 cubic feet.

This simple relation, however, applies only to the sharp-edge orifice and does not apply to orifices of the channel type, the reasons for which will appear later. It is also understood that for the above conditions the barometric pressure and gas temperature were the same.

Fig. 4 shows the agreement between the observed values for five different pressures and values obtained by using the relation

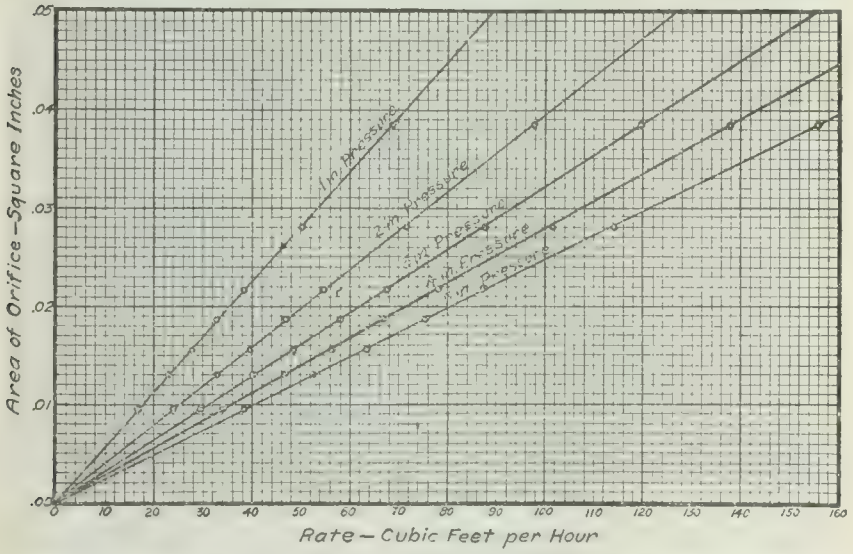


FIG. 4.—*Rates of flow of a 0.65 specific gravity gas through various sizes of orifices of type No. 1 (Fig. 6), with pressures of 1, 2, 3, 4, and 5 inches of water, respectively*

given above. For example, the orifice that passed 69 cubic feet at 1 inch pressure passed 138 cubic feet at 4 inches pressure. The No. 1 type of orifice (see Fig. 6) with gas of 0.65 specific gravity was used in this test.

(*b*) EFFECT OF SPECIFIC GRAVITY ON RATE OF FLOW OF GAS

With the same orifice pressure a light gas, such as coal gas, will flow through an orifice with greater velocity than a heavy gas, such as water gas. For a given pressure the rate of flow will vary inversely as the square root of the density as shown by the simple equation of velocity $V = 66.34 \sqrt{\frac{H}{d}}$.

In Fig. 5 are given the results obtained with three different specific gravities of gas. The tests were made with an orifice pressure of 4 inches of water and with orifices of the No. 4 type. These tests were made on different days and the temperature and barometric pressure were slightly different. For this figure and in all the other values which have been reported the gas rates have been corrected to the usual standard conditions of 60° F and 30 inches of mercury pressure.

For example, if an orifice passed 100 cubic feet of gas per hour, of 0.65 specific gravity, the same orifice, with the same pressure,

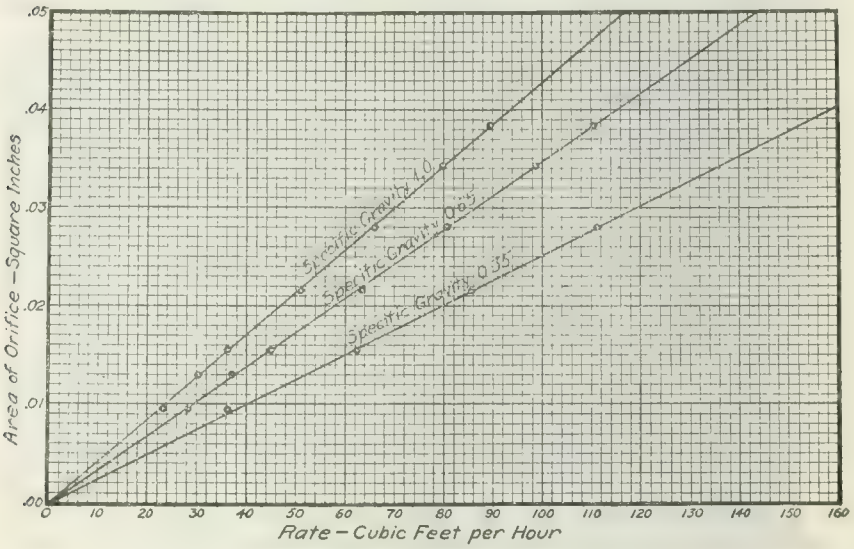


FIG. 5.—*Rates of flow, at 4 inches water pressure, of gases of 1.0, 0.65, and 0.35 specific gravity through orifices of type No. 4, Fig. 6*

would pass 136.3 cubic feet of gas per hour, of 0.35 specific gravity.

$$100 \times \sqrt{\frac{0.65}{0.35}} = 100 \times 1.363 = 136.3 \text{ cubic feet per hour.}$$

The curves in Fig. 5 give 138 cubic feet per hour, of 0.35 specific gravity, which is within about 1 per cent of the calculated value given above and is sufficiently accurate for practical purposes.

Some of the slight discrepancies of the relative gas rates are accounted for by the fact that the corrected value represents the volume of the gas that passed through the orifice, when corrected to the standard condition. The volume that would have passed through the orifice if the temperature and barometric pressure

had been actually 60° F and 30 inches of mercury pressure would be slightly different on account of the difference in density.

The change in specific gravity with any one type of gas is rarely sufficient to cause any difficulty in appliances, but where mixtures of two or more gases are being distributed, it becomes of considerable importance. The effect of specific gravity will be discussed further under air entrainment and under Section VII, "The burner tube and burner ports."

2. REASON FOR ORIFICE INVESTIGATION

Most of the experiments on flow of air through orifices, as reported by various investigators, have been made with orifices larger than those commonly used in gas appliances and with pressures which were considerably greater, and for that reason very little of the published data is applicable to this problem.

In the Gas Engineering Journal of December 1, 1917, Riley and Wilson give some values for the flow of gas of 0.550 specific gravity through orifices of various sizes. Calculating a few of these values, the discharge coefficient was found to be about 0.908. In the N. C. G. A. Practical Gas Educational Courses the tables of rates of flow through orifices with water gas of 0.65 specific gravity give a discharge coefficient of 0.624. The reason for the wide difference in these values is not evident from the published data.

In view of the wide difference in these values of the discharge coefficient, as indicated by the above data, it was considered advisable to make up various types of orifices, and to determine their relative rates of flow and their discharge coefficients. It was also thought to be of great importance to determine whether the different types of orifices would give the same relative air entrainment, since it seemed probable that with different types of orifices the shape of the gas stream would be somwehat different, and the loss of energy of the gas stream in passing through the orifice might be different, both of which would affect the air entrainment.

In the first part of this investigation the common type of orifice which has been designated as the "channel" type was used. The approach was made with a large-size drill of standard form, with an angle of approach of about 59°. To obtain orifices of different rates of flow drills of different sizes were used. An effort was made to make the channel of each orifice about one-tenth inch long. It seemed impossible, however, by this method

to make two orifices of the same size that would have the same rate of flow.

It was observed shortly after this investigation was started that the gas stream must be directed exactly into the center of the mixing tube, otherwise the air injection would be reduced. One orifice in particular did not inject the air that it should; yet to the eye it appeared to be just like the others in every respect. Its inconsistency was not understood until a stream of water was forced through it. This showed that the stream was directed off center several degrees and explained why there was a large loss in air injection. No trouble of this kind was experienced with orifices cut accurately on a lathe.

Due to the large difference in rates of flow and the difference in air injection of the various types of orifices, it was thought necessary to investigate this phase of the problem quite thoroughly before taking up the investigation of the burner itself. So far

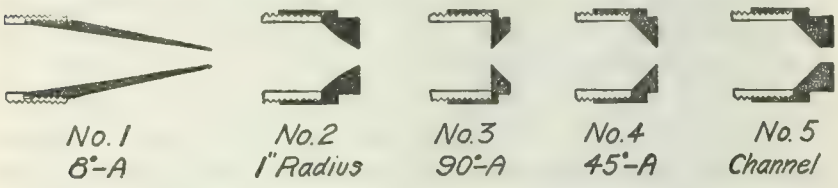


FIG. 6.—*Design of orifices. Orifices Nos. 1, 2, 3, and 4 are designated as "sharp edge" type, orifice No. 5 as "channel" type*

as we know a careful study of the discharge coefficient of various types of gas orifices and the relative air injection of each has never before been made.

3. TYPES OF ORIFICES INVESTIGATED

In Fig. 6 are shown five of the different types of orifices investigated. No. 1 has an 8° angle of approach; No. 2 was cut with a special tool that made the wall of the approach a curve of 1 inch radius; No. 3 is a sharp-edge orifice with a 90° approach and a 45° outlet; No. 4 has a 45° approach. These four types of orifices have been designated as sharp-edge orifices. No. 5 has a 45° approach and a channel of varying length as described in the text. This type of orifice we have called the channel type. Modifications of No. 5 are most commonly used.

(*a*) THE CHANNEL ORIFICE

The most common type of orifice and one that is used in most gas appliances is the type having a channel or short tube. In this class will be included all gradations from a sharp-edge orifice

to one having a channel or a tube, such as would be formed when an ordinary plug is drilled for an orifice.

The channel type of orifice may have a channel of varying length, and it may be made with different lines or angles of approach; changing either will cause a large variation in the rate of flow.

To show the effect of length of channel, orifices with a 45° angle of approach of the type shown by No. 5 in Fig. 6 were used. The orifices were made the size of No. 2 drill which has a diameter of 0.221 inch and an area of 0.0384 square inch.

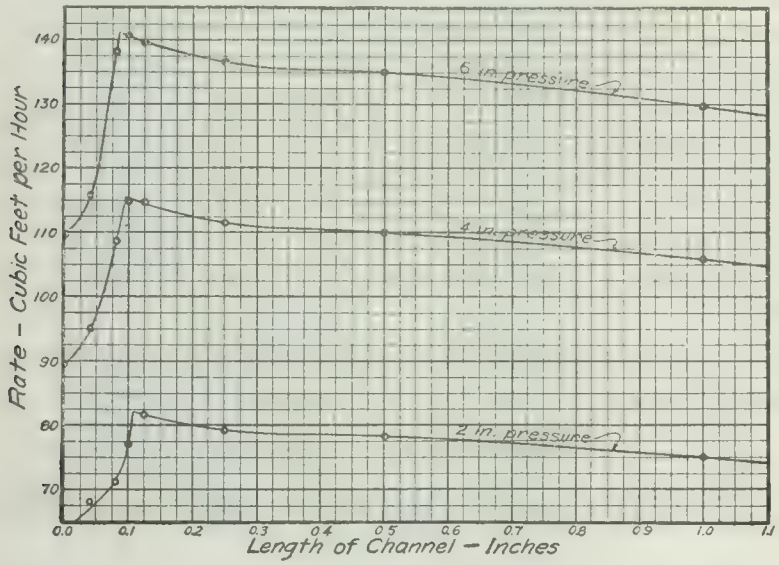


FIG. 7.—*Rates of flow of 1.0 specific gravity gas, at 2, 4, and 6 inches water pressure, through channel orifices of type No. 5, Fig. 6, with a 45° angle of approach and varying length of channel. Size of orifices, No. 2 drill*

Fig. 7 shows the very interesting characteristics of such an orifice and explains why in the preliminary investigation it was impossible to get the gas rates at the different pressures to agree with the simple theory which it seemed should apply. A study of this figure, the values of which were plotted from Table 1, shows that as the length of channel is increased, beginning with a sharp edge, the gas rate rapidly increases and reaches a maximum at a certain length of channel and then gradually decreases. It should be noted, however, that with the higher pressure the gas rate increases more rapidly than it does at the lower pressure,

and the length of channel that gives the maximum rate is not quite the same for lower pressures as it is for the higher pressures; for example, with the 0.1 inch length of channel, at 2 inches orifice pressure, the gas rate is below the maximum point, and at 6 inches the same orifice gives a gas rate which is slightly beyond the maximum rate. In other words, the length of channel which will give the maximum rate is slightly different for different pressures.

The reason for the rapid increase in rate of flow when a short-channel orifice replaces the sharp-edge orifice is that with the short channel the constriction of the gas stream, which occurs in the case of the sharp-edge orifice with gases as well as with liquids, is reduced, and although the maximum velocity may be less, the average velocity at the orifice (and therefore the volume of discharge) is increased, the rate of increase being shown by the rising curves of Fig. 7. At a certain point this effect is a maximum, and as the length of the channel is further increased the friction increases and reduces the flow. The length of tube for maximum flow varies slightly with the pressure, as shown by the curves.

TABLE 1.—Rate of Flow and Discharge Coefficient of Channel Orifices. (See Fig. 7)

[Specific gravity of gas, 1.0; size of orifice, No. 2 drill (0.221 inch diameter); orifices with 45° approach]

Length of channel	At 2-inch pressure		At 4-inch pressure		At 6-inch pressure	
	Rate of flow	Discharge coefficient (K)	Rate of flow	Discharge coefficient (K)	Rate of flow	Discharge coefficient (K)
Inch	Foot³/hour		Foot³/hour		Foot³/hour	
0.00	63.6	0.705	89.5	0.703	109.6	0.703
.04	68.5	.760	95.1	.747	115.6	.741
.08	71.3	.792	108.6	.853	138.1	.886
.10	77.1	.855	114.9	.903	140.6	.902
.125	81.6	.906	114.6	.901	139.2	.893
.25	79.2	.879	111.4	.875	136.4	.875
.50	78.2	.868	110.0	.865	135.0	.865
1.00	75.0	.832	105.7	.831	129.7	.832

It will be noted in Table 1 that the orifice of 0.08 inch length of channel has a discharge coefficient at 2 inches pressure of 0.792; at 4 inches, 0.853; and at 6 inches, 0.886, although for other lengths the differences due to the variation of pressure are less. It follows, then, that it is impossible to design an orifice of the channel type within the critical length—namely, 0.00 to about 0.125 inch—that will give a perfectly constant discharge coefficient, although

the small variations with pressure would probably not be objectionable in practice. No measurements have as yet been made of other sizes of channel orifices. The effect of length of channel on the gas rate will undoubtedly be similar, but the location of the maximum point may vary somewhat with the size.

(*b*) THE SHARP-EDGE ORIFICE

The tests with the channel orifice gave such interesting results that it seemed desirable to investigate various forms of the sharp-edge type. Fig. 8 shows the rates of flow of four types of sharp-edge orifices at 4 inches pressure with gas of 0.65 specific gravity.

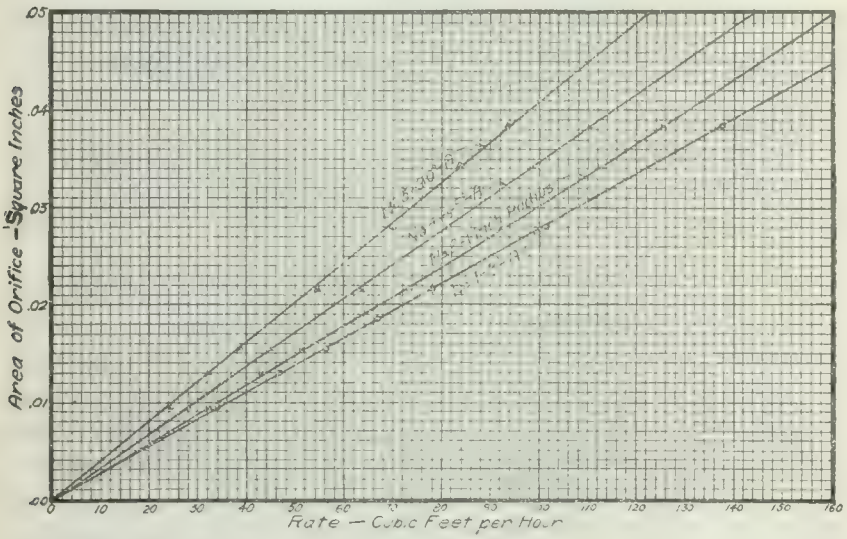


FIG. 8.—*Rates of flow of 0.65 specific gravity gas, at 4 inches water pressure, through various sizes of sharp edge orifices of types Nos. 1, 2, 3, and 4, Fig. 6*

Orifices have been tested over a range of pressure from 1 to 6 inches and with sizes from No. 2 to No. 53 drill, and it has been found that there is a linear relation between the area of the orifice and the gas rate for orifices of types 1, 3, and 4. (See Fig. 6.) From this it follows that the discharge coefficient for each one is constant over this range of sizes, although the discharge coefficient is considerably different for each type.

The "1-inch radius" type shows a slight curve, which is accounted for by the fact that all the orifices of this type were made with the same tool and the angle of approach at the immediate point of discharge was different for each size of orifice. The discharge coefficient for this type of orifice is, therefore, not a constant.

Since it had been found that the length of channel was important, it seemed desirable to investigate the effect of the angle of approach with the sharp-edge type. In Fig. 9*a* the rates of flow of orifices of the size of No. 2 drill have been shown for 2 and 4 inch pressures for seven different angles of approach. Tests are shown with two different specific gravities of gas. It is apparent from this figure that the rate of flow of the orifice varies greatly with different angles of approach.

From this figure the discharge coefficient (K) for the seven different orifices has been computed. The discharge coefficient varies from about 0.605 for a 90° approach to about 0.875 for an 8° approach. Obviously these values are related to the difference in the constriction of the gas stream, which is large with a 90° approach and very small with an 8° approach. These values have been plotted in Fig. 9*b*.

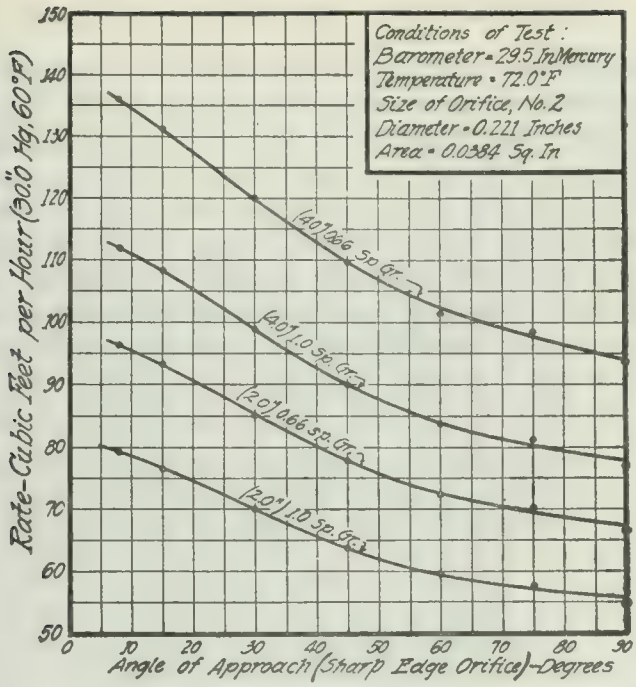
Fig. 9*b* shows that the difference in the discharge coefficient for gases of different specific gravities is exceedingly small, the two gases being air and water gas.

4. LOSS OF AIR INJECTION WITH DIFFERENT TYPES OF ORIFICES

It has been found that for all orifices of the sharp-edge type for a constant pressure at the orifice there is a linear relation between the volume of gas entering the burner and the pressure of the mixture of gas and air produced within the burner. Proof of this is shown by the results plotted in Fig. 10. When the pressure in the same burner obtained with the different types and sizes of sharp-edge orifices was plotted against their respective gas rates, the points all fell on the same straight line. It follows, then, that if the orifices were of such a size that each delivered the same quantity of gas per hour under the same orifice pressure—that is, each had the same rate of flow—the pressure in the burner would be the same with each type of orifice and the volume of air injected with each would consequently be the same.

Orifices of the channel type invariably produced a pressure in the burner which was less than that produced by the sharp-edge type, when the gas rate of the two types was the same and was produced by the same orifice pressure. This shows that the air injected by the channel orifice is always less than by the sharp-edge type, with the same pressure and gas flow. This is because the maximum velocity of the gas, and therefore its momentum, is greater with the sharp-edge orifice than with the channel orifice, and hence its air-entraining power is greater.

A


Conditions of Test:
Barometer = 29.5 In. Mercury
Temperature = 72.0° F
Size of Orifice, No. 2
Diameter = 0.221 Inches
Area = 0.0384 Sq. In
[4.0"] 0.66 Sp. Gr.
[4.0"] 1.0 Sp. Gr.
[2.0"] 0.66 Sp. Gr.
[2.0"] 1.0 Sp. Gr.
Rate-Cubic Feet per Hour (30.0" Hg. 60° F)
Angle of Approach (Sharp Edge Orifice)-Degrees

B

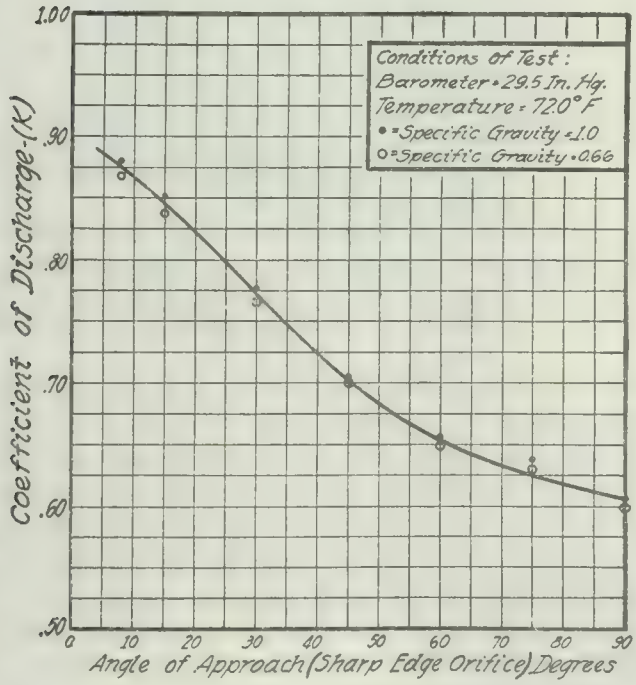

Conditions of Test:
Barometer = 29.5 In. Hg.
Temperature = 72.0° F
• = Specific Gravity = 1.0
○ = Specific Gravity = 0.66
Coefficient of Discharge-(K)
Angle of Approach (Sharp Edge Orifice) Degrees

FIG. 9.—*Graphs of discharge rates for sharp edge orifice*

A. Rates of flow of 1.0 and 0.66 specific gravity gas, at 2 and 4 inches water pressure, through sharp edge orifices, No. 2 drill size, with different angles of approach

B. Discharge coefficients of sharp edge orifices with different angles of approach

When the rate of discharge of the orifices with different lengths of channel was determined, the pressure in the burner was recorded in each case. The pressure in the burner obtained with the different channel orifices when the gas pressure was 4 inches has been plotted in Fig. 11*a*. It will be seen that the pressure in the burner rapidly increases for the short lengths of channel up to 0.1 inch and then gradually decreases. Since the gas rate changed with the different lengths of channel as well as the pressure in the burner, the relative air injection of the different orifices is therefore not apparent from the values shown in Fig. 11*a*. To get comparative data of the relative air injection of the differ-

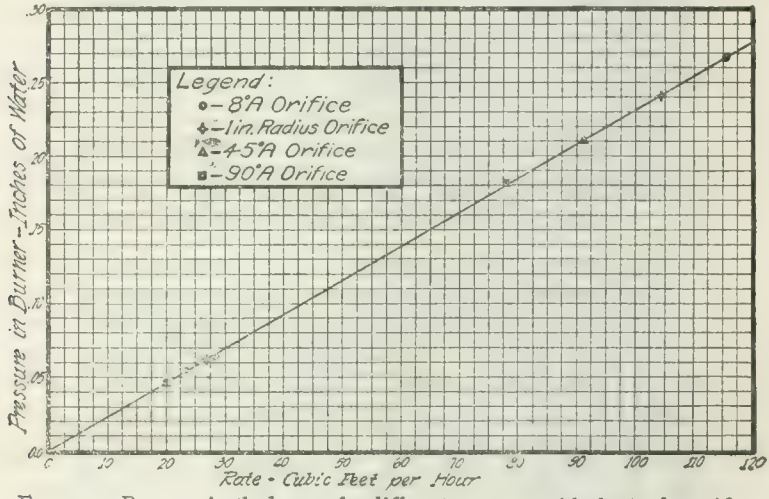


FIG. 10.—*Pressure in the burner for different gas rates with sharp edge orifices*

For these orifices any given gas rate gives the same pressure in the burner, thus indicating that the air injected is the same in each case. The pressure in the burner at the different gas rates plots on the same straight line, which shows that for sharp edge orifices there is a constant ratio between the gas rate and the pressure in the given burner and, therefore, the resistance to the flow of the gas is proportional to the gas rate

ent orifices it is necessary to compare the orifices if each delivered the same volume of gas per hour.

Fig. 11*b* has been prepared to show the relative air injection of the different lengths of channel if the orifices were of such a size that each delivered 100 cubic feet of gas per hour when under a pressure of 4 inches. This figure shows that if a sharp-edge orifice delivers 100 cubic feet of gas per hour under a pressure of 4 inches and injects 400 cubic feet of air per hour into a given burner, there is a loss of air injection with channel orifices, and that a channel orifice 0.1 inch long, for example, will inject only 381 cubic feet of air per hour under the same conditions.

It will be observed from this figure that the loss of air injection increased rapidly for the short lengths up to 0.1 inch and for further increase of length of channel the loss is not so marked.

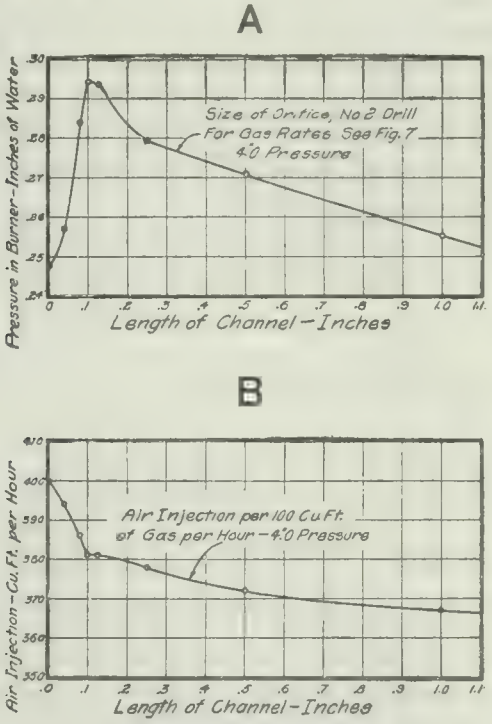


FIG. 11.—*Graphs of pressure and air injection for different lengths of channel*

A. Pressures developed in burner by 1.0 specific gravity gas at 4 inches water pressure flowing through No. 2 drill size orifices with different lengths of channel

B. Air injected by gas at 4 inches water pressure flowing at the rate of 100 cu. ft. per hour from No. 2 drill size orifices with different lengths of channel

5. HOW TO CALCULATE THE RATE OF FLOW AND SIZE OF GAS ORIFICES

If the rate of flow of an orifice is desired, or the size of an orifice is required for any given gas rate, it is necessary to know the gas pressure, the specific gravity of the gas, and the discharge coefficient of the orifice.

The N. C. G. A. educational course on industrial fuel, in lesson paper No. 5, Fig. 12, gives a chart for computing the sizes and rates of flow of orifices; but this chart is not of very great value unless one knows the discharge coefficient for the type of orifice that is to be used. In the same lesson paper there is also given a table of rates of flow for "thin orifices, 0.22 inch thick." We

have previously stated that from this table of rates of flow of orifices the discharge coefficient was found to be about 0.624.

From the data and curves given in the preceding pages of this paper it should be possible to estimate very closely the discharge coefficient for any type of orifice, and with that information it should be possible to calculate the rate of flow and size of orifices.

A simplified formula for calculating the rate of flow of an orifice in terms of its area and the discharge coefficient, and of the density and pressure of the gas, can be deduced as follows:

Let q = rate of flow of gas from orifice in cubic feet per second;

a = area of orifice in square feet;

V = theoretical velocity of gas in feet per second;

K = orifice constant, or discharge coefficient; then

$$q = aKV \text{ or } a = \frac{q}{KV}$$

It has previously been shown that

$$V = 66.34\sqrt{\frac{H}{d}}, \text{ where}$$

H = orifice pressure in inches of water, and

d = specific gravity of gas (air = 1.0).

Therefore, the rate of discharge

$$q = aKV = 66.34\ aK\sqrt{\frac{H}{d}} \text{ cubic feet per second.}$$

This formula can be put in a more convenient form as follows:

Let Q = rate of discharge of orifice in cubic feet per hour, and

A = area of orifice in square inches; then

$$Q = 3600 \times 66.34\ \frac{A}{144}\ K\sqrt{\frac{H}{d}}$$

$$= 1658.5\ AK\sqrt{\frac{H}{d}}; \text{ or}$$

$$A = \frac{Q}{1658.5\ K\sqrt{\frac{d}{H}}}$$

In this form the formula will enable one to calculate readily any one unknown value by substituting in the formula the known values.

V. PRINCIPLES GOVERNING AIR INJECTION

1. AIR ENTRAINMENT AND MOMENTUM OF THE GAS STREAM

The momentum of a body is equal to its mass times its velocity. This is true with gases as well as solids and liquids. In an atmospheric burner a part of the momentum of the gas stream is com-

municated to the air; therefore the quantity of air injected will depend upon the velocity of the gas through the orifice, the rate of flow, and the specific gravity of the gas. There is, of course, always some loss in momentum due to friction and eddy currents, and the total momentum of the mixture is, therefore, less than the momentum of the gas stream at the orifice.

How much of the momentum of the gas stream is converted into the momentum of the mixture will depend upon the design of the injecting tube and the resistance of the burner. In general, for any one burner, the injecting tube that will produce the largest pressure in the burner for a given pressure at the orifice will be the most efficient air injector. The resistance of the burner will depend upon the shape and size of the burner and the area of the ports.

The numerous experiments have conclusively demonstrated that if a given rate of flow of gas issuing from an orifice under a pressure of 2 inches, for example, will inject two volumes of air to one of gas into a given burner, and the gas pressure is increased, thereby increasing the gas rate, the volume of air injected will be increased, but the air-gas ratio will remain the same as it was before the change of gas pressure.

To show the effect of changing the gas pressure, specific gravity of the gas, and the gas rate on air entrainment, there are given below several problems with values taken from the experimental data.

2. RELATION BETWEEN THE MOMENTUM OF THE GAS STREAM AND THE MOMENTUM OF THE STREAM OF THE MIXTURE

The tests of the different pipe burners have shown that for any given burner operated over the range of gas pressure from 2 to 6 inches the ratio of the momentum of the gas stream to the momentum of the stream of the air-gas mixture is practically a constant, irrespective of (1) a change of gas pressure, (2) a change of gas rate, and (3) a change of specific gravity of gas. Then if

m = mass of gas per second that issues from an orifice,

M = mass of air per second that is injected into a given burner,

V = theoretical velocity of gas stream in feet per second = $\sqrt{2gh}$,

v = velocity at first port of the air and gas mixture in feet per second,

mV = the momentum of gas stream, and

$(M+m)v$ = the momentum of mixture of air and gas at first port, therefore, as stated above,

$$\frac{mV}{(M+m)v} = \text{a constant ratio for any given burner.}$$

A simplified formula can be developed as follows, which will greatly aid the reader to apply this useful information in calculating practical problems: Let

q = gas rate in cubic feet per second = cubic feet per hour ÷ 3600;
W = mass of 1 cubic foot of air at 30 inches Hg and 60° F;
d = specific gravity of gas (air = 1.0);
r = air to gas ratio of mixture passing through burner;
x = cross-sectional area of pipe in square feet (0.01038 square foot for 1¼ inch pipe burner);
R = ratio of momentum of gas stream to momentum of stream of mixture;
V = theoretical velocity of gas stream in feet per second;
m = mass of gas stream per second = qdW;
mV = momentum of gas stream = $qdWV$ foot pounds per second;
rq = volume of air injected into the burner per second;
M = mass of air per second = rqW;
v = velocity of the mixture at first port, which is the volume of air per second in cubic feet + volume of gas per second in cubic feet ÷ cross-sectional area of pipe in square feet,

$$= \frac{rq+q}{x} = q\frac{(r+1)}{x} \text{ feet per second; and}$$

$(M+m)\,v$ = the momentum of the mixture

$$= (rqW+qdW)\frac{q\,(r+1)}{x} = \frac{qW\,(r+d)\,q\,(r+1)}{x} \text{ foot pounds per second:}$$

Therefore the ratio of momentum of gas stream to momentum of stream of mixture

$$R = \frac{mV}{(M+m)\,v} = \frac{qdWV}{\frac{qW\,(r+d)\,q\,(r+1)}{x}} = \frac{dVx}{q\,(r+d)\,(r+1)}$$

It is perhaps more convenient to use the gas rate in terms of cubic feet per hour, and the cross-sectional area of the pipe in square inches; then

$$R = \frac{3600}{Q}\,\frac{X}{144}\,\frac{dV}{(r+d)\,(r+1)} = \frac{25XdV}{Q\,(r+d)\,(r+1)}, \text{ where}$$

Q = the gas rate in cubic feet per hour, and
X = cross-sectional area of pipe in square inches = 1.496 square inches for 1¼ inch pipe burner.

Also the equation is much more readily applied if, in place of V, its value, $66.34\sqrt{\frac{H}{d}}$, is substituted, where H = the pressure of the gas in inches of water. Then

$$R = \frac{25Xd \times 66.34\sqrt{\frac{H}{d}}}{Q\ (r+d)\ (r+1)},$$

$$= \frac{1658.5\ X\ \sqrt{Hd}}{Q\ (r+d)\ (r+1)}.$$

The following examples show the ratio between the momentum of the gas stream and the momentum of the stream of the mixture in which pipe burners were used. The burners were equipped with fixed orifices. It should be evident that changing either the design of the injecting tube or the port area of the burner head, as well as a change of adjustment of the air shutter if one is used, will cause a change in the volume of air injected into the burner, therefore, a change of the ratio (R). Also a change of the type of orifice may cause a change of the momentum of the gas stream which in turn will cause a change of the ratio R. It follows that each particular burner has a constant ratio R, if no change or adjustment of the burner parts is made.

EXAMPLE NO. 1.—To illustrate that for a given burner the ratio R between the momentum of the gas stream and the momentum of the stream of the mixture is a constant when the gas rate is changed but the pressure remains constant. The gas rate and the air-gas ratio values for both of the following conditions were taken from "(4 inch) no tube ratio" curve of Fig. 26. The gas was 0.65 specific gravity. The burner was a $1\frac{1}{4}$ inch pipe burner with 1.05 square inch port area.

	Case I	Case II
Gas rate (cubic feet per hour).....(Q)	47.5	67.0
Pressure of gas (inches of water).....(H)	4.0	4.0
Specific gravity of gas.....(d)	.65	.65
Air-gas ratio of mixture.....(r)	3.72	3.00
Cross-sectional area of pipe (sq. in.)...(X)	1.496	1.496
Ratio $R = \frac{1658.5X\sqrt{Hd}}{Q(r+d)(r+1)}$	$\frac{1658.5 \times 1.496\sqrt{4.0 \times 0.65}}{47.5\ (3.72+0.65)\ (3.72+1)} = 4.08$	$\frac{1658.5 \times 1.496\sqrt{4.0 \times 0.65}}{67.0\ (3.0+0.65)\ (3.0+1)} = 4.09$

EXAMPLE NO. 2.—To illustrate that for a given burner the ratio R between the momentum of the gas stream and the momentum of the stream of the mixture is a constant when the gas pressure is

changed, but the gas rate is kept constant. These values were also taken from the "no tube ratio" curves of Fig. 26.

	Case I	Case II
Gas rate (cubic feet per hour).....(*Q*)	47.5	47.5
Pressure of gas (inches of water).....(*H*)	2.0	4.0
Specific gravity of gas.....(*d*)	.65	.65
Air-gas ratio of mixture.....(*r*)	3.00	3.72
Cross-sectional area of pipe (sq. in.).....(*X*)	1.496	1.496
Ratio $R=\frac{165\sqrt{.5}X\sqrt{Hd}}{Q(r+d)(r+1)}$	$\frac{1658.5\times1.496\sqrt{2.0\times0.65}}{47.5\ (3.0+0.65)\ (3.0+1)}=4.08$	$\frac{1658.5\times1.496\sqrt{4.0\times0.65}}{47.5\ (3.72+0.65)\ (3.72+1)}=4.08$

EXAMPLE NO. 3.—To illustrate that for a given burner the ratio *R* between the momentum of the gas stream and the momentum of the stream of the mixture is a constant when the specific gravity

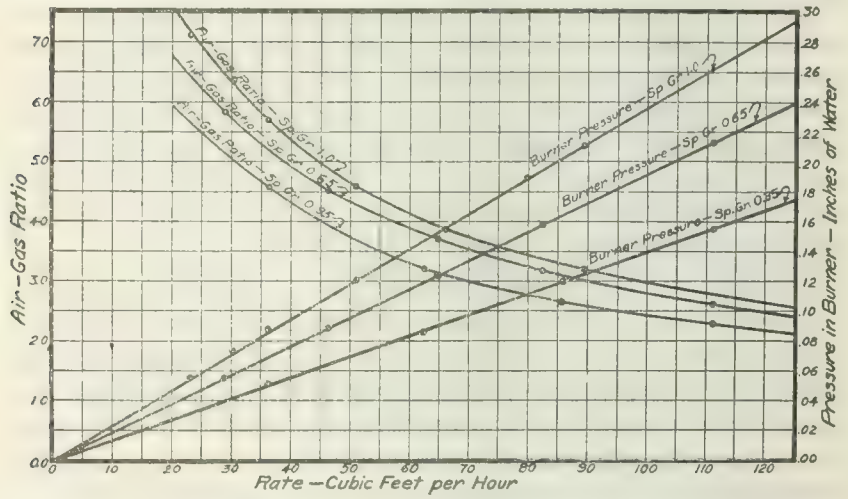


FIG. 12.—*Air injection and pressure in burner obtained with 1.0, 0.65, and 0.35 specific gravity gases flowing through sharp edge orifices at 4 inches water pressure*

Burner used is a 1¼-inch pipe burner of 0.75 square inch port area fitted to improved injector

of the gas is changed. The values are taken from the 0.35 and 0.65 specific gravity of gas curves of Fig. 12.

	Case I	Case II
Gas rate (cubic feet per hour).....(*Q*)	45	45
Pressure of gas (inches of water).....(*H*)	4.0	4.0
Specific gravity of gas.....(*d*)	.35	.65
Air-gas ratio of mixture.....(*r*)	4.0	4.6
Cross-sectional area of pipe (sq. in.).....(*X*)	1.496	1.496
Ratio $R=\frac{1658.5X\sqrt{Hd}}{Q(r+d)(r+1)}$	$\frac{1658.5\times1.496\sqrt{4.0\times0.35}}{45.0\ (4.0+0.35)\ (4.0+1)}=3.00$	$\frac{1658.5\times1.496\sqrt{4.0\times0.65}}{45.0\ (4.6+0.65)\ (4.6+1)}=3.02$

3. HOW TO CALCULATE THE AIR-GAS RATIO (r) OF A GIVEN BURNER

It should be evident that if the ratio R of the momentum of the gas stream to the momentum of the stream of the mixture is known for a given burner, it is possible from the equation which has been developed to solve for any one unknown value. In the equation $R = \frac{1658.5 X \sqrt{Hd}}{Q(r+d)(r+1)}$, probably the most important unknown value to solve would be the air-gas ratio (r). Solving the above equation for air-gas ratio

$$r = \sqrt{\frac{1658.5 X \sqrt{Hd}}{RQ} + 0.25d^2 - 0.5d + 0.25} - 0.5d - 0.5$$

It has been shown that R for the burner used to determine the "no tube ratio" curves of Fig. 26 has a value of 4.08. Suppose it is desired to know air-gas ratio (r) where the gas pressure $H =$ 2.0 inches, and the rate of flow $Q = 50.0$ cubic feet per hour. The specific gravity of the gas $d = 0.65$, and the cross-sectional area of the pipe burner $X = 1.496$ square inches. Then,

$$r = \sqrt{\frac{1658.5 \times 1.496 \sqrt{2.0 \times 0.65}}{4.08 \times 50.0} + 0.25\ (0.65)^2 - (0.5 \times 0.65) + 0.25} - (0.5 \times 0.65) - 0.5 = 2.91$$

This agrees with the air-gas ratio shown by the "(2-inch) no tube ratio" curve of Fig. 26, which was obtained with 2 inches pressure and the above burner.

4. RELATION BETWEEN GAS PRESSURE AND MOMENTUM OF GAS STREAM

From the equation $V = 66.34\sqrt{\frac{H}{d}}$ it is seen that the velocity of a gas stream varies as the square root of the gas pressure, and since the mass issuing from the orifice varies as the velocity of the gas stream, the momentum of the gas stream is directly proportional to the pressure.

For example, let us assume that at 1-inch pressure the volume of gas of d specific gravity issuing from an orifice is q cubic feet per second, and the velocity of the stream is V feet per second. The momentum of the gas stream will be mass $\times$ velocity $= (qdW)\ V$ foot pounds per second, where W = the mass of a cubic foot of air at 30.0 inches Hg and 60° F.

If the pressure is now increased to 4 inches the velocity will be doubled, therefore the volume going through per second will be doubled, which doubles the mass per second, and the momentum will be

$$(2qdW)2V = 4(qdW)V \text{ foot pounds per second.}$$

5. EFFECT OF CHANGE OF SPECIFIC GRAVITY ON THE MOMENTUM OF THE GAS STREAM

If two orifices are of such a size that when under the same pressure each delivers the same volume of gas per hour and the gases are of different specific gravity, the momenta of the two gas streams are proportional to the square roots of their gravities; for example, compare gases of 0.35 and 0.65 specific gravities issuing from different orifices under the same pressure. If q cubic feet per second issues from each under 1-inch pressure, the momentum of the 0.65 specific gravity gas will be $qdWV = q \times 0.65 \times W \times 82.3 = 53.5\ qW$ foot pounds per second, where $V = 82.3$ feet per second.

The velocity of the 0.35 specific gravity gas will be $82.3 \times \sqrt{\frac{0.65}{0.35}} = 112.1$ feet per second.

The momentum of the 0.35 specific gravity gas will be $q \times 0.35 \times W \times 112.1 = 39.25\ qW$ foot pounds per second and $\sqrt{\frac{0.65}{0.35}}$ is proportional to $\frac{53.5\ qW}{39.25\ qW}$.

6. RELATION BETWEEN MOMENTUM OF MIXTURE AND PRESSURE IN THE BURNER

It has been proved that there is a definite ratio between the momentum of the gas stream and the momentum of the stream of the mixture. For any increase in the gas rate there is a corresponding increase in the momentum of the stream of the mixture, resulting in an increase in the pressure in the burner. Fig. 12 shows the pressures in the burner with three specific gravities of gas, and Figs. 21 to 27, inclusive, show the pressures in different burners with different gas rates and gas pressures.

7. THE VARIOUS RELATIONS SUMMARIZED

From the fundamental theory and the relations which have been illustrated in the preceding examples it is possible to summarize the most important relations as follows:

1. The ratio between the momentum of the gas stream and the momentum of the stream of the mixture is always the same for a burner of a given design irrespective of orifice pressure, specific gravity of gas, or the volume of the air and gas mixture going through the burner.

2. Where the gas rate is increased by change of pressure, the momenta of the gas streams are directly proportional to the pressures.

3. When the same volumes of gases of different specific gravities issue from different orifices under the same orifice pressure the momenta of the gas streams are proportional to the square roots of the specific gravities of the gases.

4. When the pressure is changed to give the same gas rate for gases of different specific gravity, the air entrainment is proportional to the specific gravities.

5. The pressure at any one point in the burner increases in direct proportion to the increase in the momentum of the stream of the mixture.

If a change in gas pressure or gas rate should cause a considerable change in the temperature of the burner, the volume of air injected would be slightly different from the calculated values for the new condition, and there would not be quite the agreement between the above stated relations.

VI. THE INJECTING TUBE

1. THE ADVANTAGE OF AN INJECTING TUBE

When a gas under pressure issues from an orifice, the momentum of the gas causes the surrounding air to be set in motion in the direction of the gas stream. Because of the great velocity of the gas stream eddy currents are set up in the surrounding air. This eddy-current motion consumes much of the energy of the gas.

The function of an injector is to eliminate, as much as possible, this eddy-current movement, and to give a ratio of air to gas suitable for the given purpose.

The idea of a proper constriction in the mixing tube of the burner in order to secure better air injection is not new. In fact nearly all domestic range burners are so designed, although exact information as to the best form of injector for gas burners has been wanting.

In atmospheric burners the flame characteristics of combustible gases vary with the change of ratio of primary air to gas. When

the air-gas ratio of the mixture entering the burner is low, the combustion at the ports is slow, the flame is long, and the heat liberated per unit of flame area is low, relative to that of a mixture of a higher air-gas ratio. When the air-gas ratio of the mixture entering the burner is increased, the rate of combustion at the ports is increased, the flame height is decreased, the flame area is decreased, and the heat liberated per unit of flame area is increased. The higher air-gas ratio condition permits of greater concentration of heat, and, therefore, greater rate of consumption per unit of burner surface.

By the use of a correctly designed injecting tube it is possible to secure—(1) increased efficiency, resulting from higher flame temperature, and (2) increased rate of consumption, resulting from the injection of a greater quantity of air.

There is no better demonstration of the practical value of the use of an injector than that made by Thompson King, of the Consolidated Gas & Electric Co., Baltimore, Md., during the extremely cold winter of 1917–18. Many installations of gas-burning house-heating boilers in that city were entirely inadequate to supply the required heat during the severe weather, and in order to secure more heat from the boilers it was necessary to increase the gas consumption of the burners. This was accomplished by inserting injecting tubes made out of tin into the mixing tubes of the burners, whereby the rate of consumption of the boilers was increased sufficiently to give entirely satisfactory results.

2. INJECTOR DESIGN AND POSITION OF ORIFICE

Bearing in mind the advantages of an injector, it was decided to determine, if possible, what design of injector would cause the maximum air injection.

To eliminate the effect of as many variables as possible, all of the preliminary investigations were made with one burner, and the design of the injector was varied to determine the most efficient injector for that burner. The burner selected was a 1¼-inch pipe burner with a port area of 0.75 square inch. Since the specific gravity of the gas does not affect the design of the injector, air instead of combustible gas was used for the sake of convenience.

By using a constant orifice pressure the gas rate was kept constant. Observing the pressure obtained in the burner by the use of different injectors, the relative merits of different designs of injecting tubes could be determined. It is evident that with a

constant gas rate the injector which gives the maximum pressure in the burner must necessarily inject the most air.

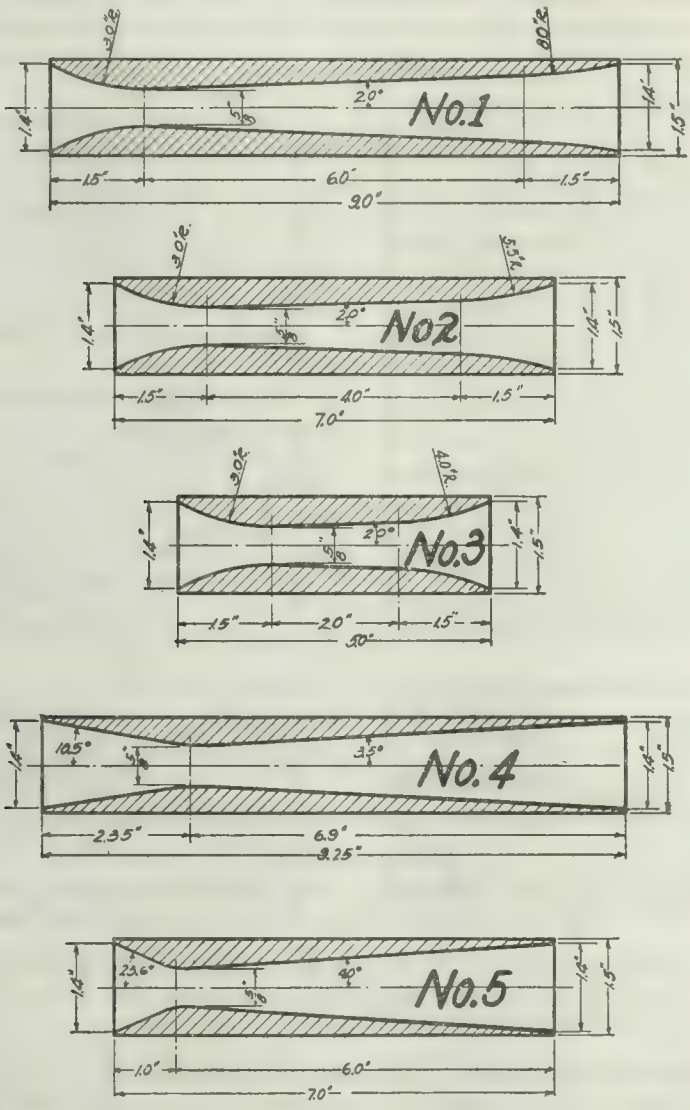


FIG. 13.—*Section views showing dimensions of five designs of injecting tubes*

In Fig. 13 and Table 2 are shown the dimensions of some of the injecting tubes that were made and tested. These injectors were made of brass and cut on a lathe. The end of the pipe burner, designated as the mixing tube, was reamed out, so that the injecting tube made a tight fit when inserted in the burner. Tube

No. 5 is the type of injector which was used in the first part of the investigation. It was soon realized that, in order to draw definite conclusions in regard to the design of the optimum injector, it was necessary to investigate the shape and length of the approach, the area of the throat, as well as the shape and length of the outlet.

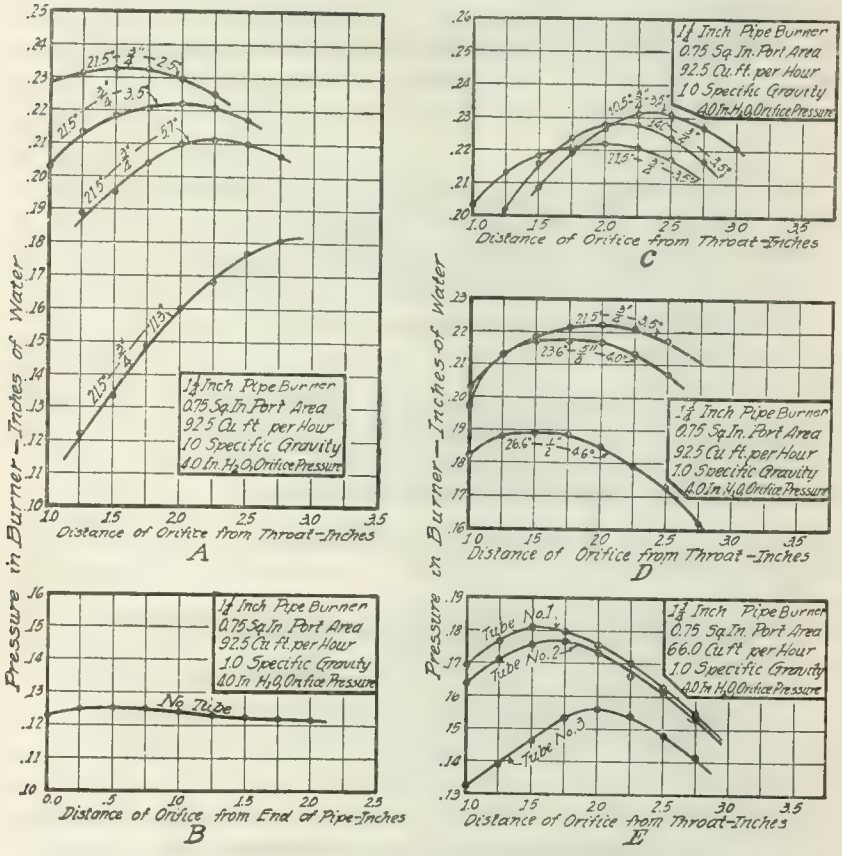


FIG. 14.—*Pressures developed in a 1¼-inch pipe burner, 0.75 square inch port area with 1.0 specific gravity gas flowing through a sharp edge orifice at 4 inches water pressure using different designs of injecting tubes*

In a previous section of this report it has been shown that there is a linear relation between the pressure in the burner and the gas rate in cubic feet per hour when the orifice pressure is constant. This pressure in the burner is known as the maximum pressure and was obtained by changing the distance of the orifice from the throat of the injector. The distance of the orifice from the throat is extremely important, and it seems that the best position of the orifice for any given injector can be determined only by experiment.

The curves of Fig. 14 will show fairly definitely the relation between the position of the orifice and the design of the injector. Once this position of maximum injection is obtained, it will hold good for that injector with any pressure at the orifice and with any gas rate.

The charts of Fig. 14 show the development of the investigation of the injector design. Chart B of this figure shows that the position of the orifice relative to the end of the burner is not so important when an unimproved injector is used. If the gas rate is kept constant at 92.5 cubic feet per hour and the different injecting tubes used in determining the curves of chart A are inserted into the burner, the advantage of an improved injector is clearly shown. These curves show the pressures obtained by the use of four injectors where the throat diameter and inlet angle were the same in each case, each injector having, however, a different outlet angle. The injector used to obtain the upper curve of this chart had an outlet length of 8 inches, while the outlet length of the injector producing the lower curve was 2 inches. It is interesting to note the positions of the orifice at which the maximum pressures for these two injectors were obtained. In the case of the former it was 1½ inches, while that of the latter was at least 3 inches. These curves contrast sharply the comparative values of long and short outlets obtained by changing the outlet angle, the longer outlet, which has the smaller slope, giving much better air injection.

The curves in chart C, Fig. 14, were obtained by using three injectors of the same throat diameter and outlet angle but of different inlet angle. The outlet length for these three injectors was 6 inches. The 10.5° inlet represents an inlet length of 2 inches and the 21.5° inlet represents an inlet length of 1 inch. This chart shows that the angle of approach is not so important as the outlet angle, and also that the position of the orifice for maximum injection changes as the inlet angle varies.

The curves of chart D, Fig. 14, are those of pressures obtained with three injectors, all of 1 inch length of inlet and 6-inch outlet length, in which the throat diameters were different. These curves are not of great value, as it was later found that in order to obtain the maximum injection of air in any burner there must be a definite relation between the area of the injector throat and the port area of the burner. The development of this relation is discussed under the flow capacity of an injector.

TABLE 2.—Dimensions and Designations of Various Designs of Injecting Tubes Tested in Pipe Burners of 1¼-Inch Diameter

[S=slope; R=radius]

Inlet dimension		Throat diameter	Outlet dimension		Injecting tube designation	Total length
Degrees slope or inches radius	Length		Slope angle	Length		
	Inches	Inch	Degrees	Inches		Inches
3″ R	1.5	5/8	2.0	6.0	3″ R–5/8–6″ (No. 1)	9.0
3″ R	1.5	5,8	2.0	4.0	3″ R–5/8–4″ (No. 2)	7.0
3″ R	1.5	5/8	2.0	2.0	3″ R–5/8–2″ (No. 3)	5.0
10°.5	2.40	5/8	3.5	6.85	10°.5–5/8–3°.5 (No. 4)	9.25
23°.6	1.0	5/8	4.0	6.0	23°.6–5/8–4°.0 (No. 5)	7.0
2″ R	1.30	5/8	3.5	6.85	2″ R–5/8–3°.5	8.15
2″ R	1.35	1/2	3.5	7.9	2″ R–1/2–3°.5	9.25
10°.5	2.7	1/2	3.5	7.9	10°.5–1/2–3°.5	10.6
26°.6	1.0	1/2	4.6	6.0	26°.6–1/2–4°.6	7.0
2″ R	1.25	3/4	3.5	6.0	2″ R–3/4–3°.5	7.25
10°.5	2.0	3/4	3.5	6.0	10°.5–3/4–3°.5	8.0
14°.0	1.5	3/4	3.5	6.0	14°.0–3/4–3°.5	7.5
21°.5	1.0	3/4	3.5	6.0	21°.5–3/4–3°.5	7.0
21°.5	1.0	3/4	2.5	8.0	21°.5–3/4–2°.5	9.0
21°.5	1.0	3/4	5.7	4.0	21°.5–3/4–5°.7	5.0
21°.5	1.0	3/4	11.3	2.0	21°.5–3/4–11°.3	3.0
6″ R	1.75	7/8	2.5	6.5	6″ R–7/8–2°.5	8.25
3″ R	1.4	7/8	3.5	4.85	3″ R–7/8–3°.5	6.25

3. THE OPTIMUM INJECTOR

From a study of the preceding charts, and from observations made on other injectors which it did not seem necessary to report, it is possible to draw some definite conclusions concerning the design of the optimum injector:

1. The change of the lines of the approach of the inlet to the lines of the outlet should be gradual.

2. The approach should follow approximately a curvature which should be not less than 3 inches radius for a 5/8-inch throat. Other sizes should be proportioned about the same.

3. The outlet angle should be about 2°.

It is, of course, true that in some installations the length of the injector is limited, and for that reason injectors Nos. 1, 2, and 3, shown in Fig. 13, were designed. A comparison of the results obtained with these three injectors is shown in chart E, Fig. 14. This chart shows that little is to be gained by making the 2° outlet slope longer than 4 inches, but that there is considerable dropping off in pressure in the burner if the length of the outlet slope is only 2 inches. Note that the position of the orifice giving maximum pressure in the burner changed with the variation in

outlet length. The injecting powers of these injectors are easily compared from a study of Fig. 15. Any injector of different design that for a given gas rate would give a higher pressure in the burner than that shown by No. 1 in Fig. 13 would, of course, be a better injector.

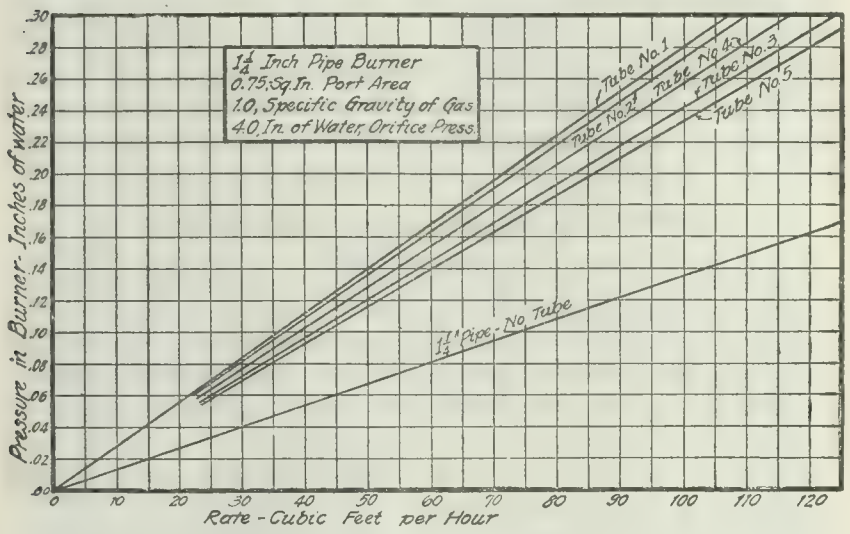

1¼ Inch Pipe Burner
0.75 Sq. In. Port Area
1.0, Specific Gravity of Gas
4.0 In. of Water, Orifice Press.
Tube No. 1
Tube No. 2
Tube No. 3
Tube No. 4
Tube No. 5
1¼ Pipe, No Tube
Pressure in Burner-Inches of water
Rate-Cubic Feet per Hour

FIG. 15.—*Relative air injection with injecting tubes of different designs as indicated by the pressure in the burner*

The injecting tubes shown in Fig. 13 were used and are compared with the same burner when a 1¼-inch pipe served as an injecting tube.

In order to show the injecting powers represented by the pressure curves in Fig. 15 the following table has been prepared:

TABLE 3.—Relative Injecting Power of Different Injectors as Indicated by the Resulting Air-Gas Ratios

Type of injecting tube	Air-gas ratio		Type of injecting tube	Air-gas ratio	
	At gas rate 50.0 cubic feet per hour	At gas rate 75.0 cubic feet per hour		At gas rate 50.0 cubic feet per hour	At gas rate 75.0 cubic feet per hour
No injecting tube	3.25	2.45	No. 3	4.55	3.50
No. 1	5.00	3.85	No. 4	4.75	3.65
No. 2	4.90	3.80	No. 5	4.40	3.40

From a study of this table we note that in the case of this particular burner with a rate of 50 cubic feet per hour it is possible to inject 54 per cent more air by the use of injecting tube No. 1 than without the use of an injector. Injecting tube No. 1 is only

about 2 per cent better as an injector than No. 2, about 9 per cent better than No. 3, 5 per cent better than No. 4, and 12 per cent better than No. 5.

From conclusions based on our study of many injectors with different designs we believe that any improvement over the type of the design of injector No. 1 will not add more than a very few per cent to the injecting power. It should be remembered, also, that these injectors are made with smooth surfaces, and that in practice, when the injector is cast, the unavoidable rough surface may offer an appreciable resistance which might cause slightly less favorable results than have been obtained in the laboratory.

4. THE FLOW CAPACITY OF AN INJECTING TUBE

The determination of the flow capacity of an injector seemed at first to be a difficult problem, but it was greatly simplified as soon as it had been determined that for any one burner and injecting tube there was a definite relation between the different variables, and that the best injector for any one burner with any one specific gravity of gas, orifice pressure, and volume of gas going through the injector was also the best injector for any other specific gravity of gas, orifice pressure, etc.

The problem resolved itself into a determination of the proper relation between the total port area of the burner and the area of the injector throat. Injectors of definite throat areas and burners with definite port areas were necessary. Three injectors of the following dimensions were made: Each had an angle of approach of 10.5°, an outlet angle of 3.5° with throat diameters of one-half, five-eighths, and three-quarters inch, respectively. This design is shown by injector No. 4, Fig. 13. Since it is essential that the change from the lines of the inlet to those of the outlet should be gradual, the injectors were cut with the throat diameters smaller than desired and then carefully rounded off so that their lines were, as nearly as possible, exactly alike.

Four pipe burners of 0.45, 0.75, 1.05, and 1.35 square inches port area, respectively, were made. The ports were made with a No. 30 drill, arranged in two rows, staggered and spaced five-eighths inch from center to center.

In this study air was again used for the sake of convenience, and an orifice pressure of 4 inches of water was used.

Since each injector has its own distance of the orifice from the throat which produces the maximum pressure in the burner, it was necessary to determine that position for each injector. To be

absolutely sure that this distance of the orifice held good for any given injector when the port area had been changed, the tests shown in Fig. 16 were made. Again, to be sure that the maximum pressure obtainable in the burner for any gas pressure bore a linear relation to the gas rate with a change of port area, numerous tests represented by the curves shown in Fig. 16 were made. Invariably when the gas rates were changed the maximum pressures fell on the same straight line for a given injector and a given port area. These maximum pressures were plotted and are shown in

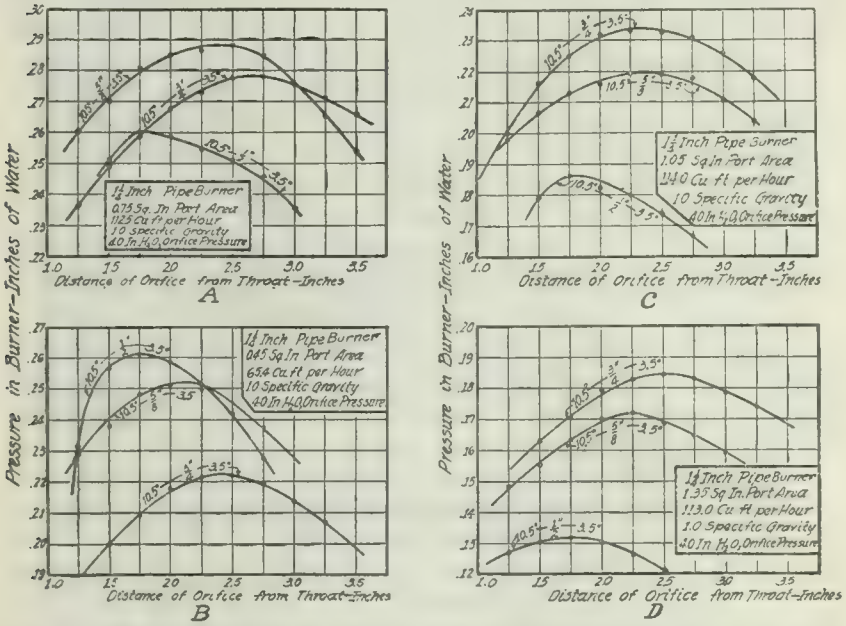


FIG. 16.—*Pressures developed in 1¼-inch pipe burners of different port areas using different injecting tubes with 1.0 specific gravity gas flowing through a sharp edge orifice at 4 inches water pressure*

Fig. 17. The curves of Fig. 17 represent the best injection that could be obtained with the three different injectors and the four burners having different port areas. It should be noted how the pressure in the burner changes with the injecting tubes of different throat diameters for a given burner and also how it varies with the port area of the burner. Of the three pressure lines shown for the burners with different port areas the injector with the throat diameter that gave the greatest pressure was, of course, the best injector. For the 0.45 square inch port area, the one-half-inch diameter throat proved to be the best injector, yet for the other burners it was the poorest. The five-eighths-inch

diameter showed up best with the 0.75 square inch port area burner, while the three-fourths-inch diameter was best for both the 1.05 and 1.35 square inch port area burners.

From the curves of Fig. 17 it is possible to draw the curves in Fig. 18; and from these latter curves we can arrive at the relation between area of the injector throat and port area of the burner. To obtain curves that would show the comparative value of the injectors with different throat diameters, it was necessary to select a constant gas rate and plot the pressure in the burner for the different diameters against the port area of the burner. This has been done for two different gas rates in Fig. 18. This figure

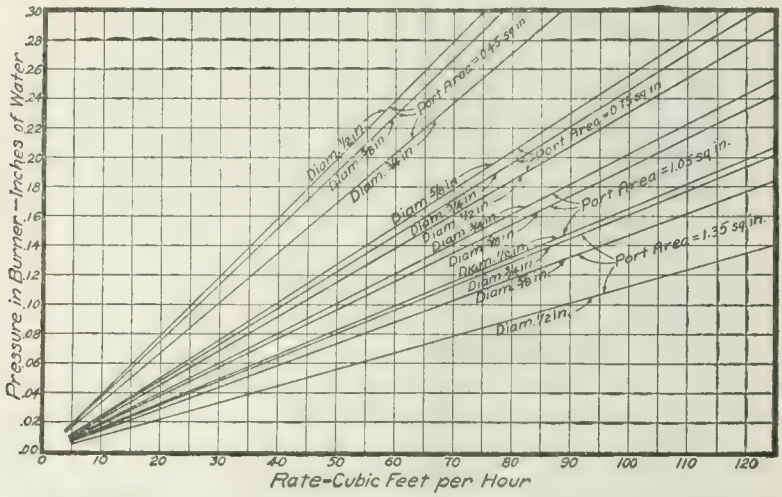


FIG. 17.—*Pressures in burners of different port areas fitted to injectors of different flow capacities*

These curves were determined from the maximum values taken from curves in which those of Fig. 16 are included. Pressures developed by gas flowing through orifice at 4 inches water pressure

indicates at a glance which diameter is the best for the different port areas. It should also be noted that the curves for the two gas rates are exactly alike, which is to be expected. A study of the curves for either gas rate shows, for instance, that the one-half and five-eighths inch throat diameter are equally good injectors for a port area of 0.55 square inch, and that the five-eighths and three-fourths inch throat diameter are equally good at 0.85 square inch port area, because in each case the pressure in the burner would be equal.

It would seem from the intersection of the curves of Fig. 18 that the five-eighths-inch throat would produce a higher pressure for a 0.70 square inch port area than that shown in the figure because

the result of the five-eighths-inch diameter for the 0.75 square inch port area is so near the intersection for the three-fourths-inch

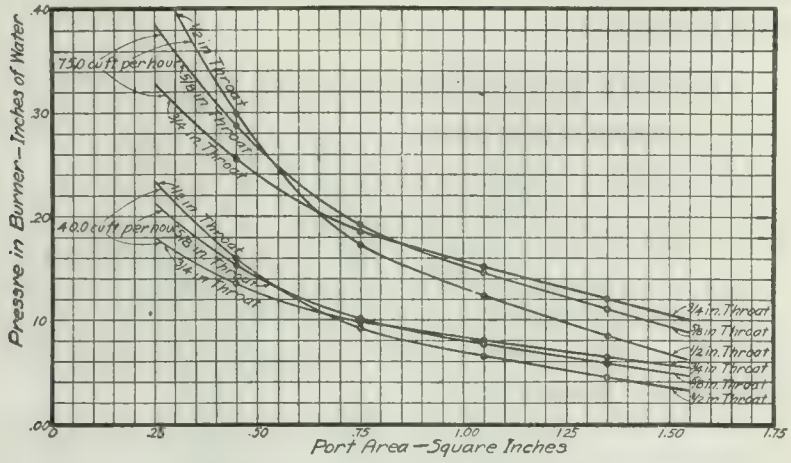


FIG. 18.—*Curves showing pressures developed in burners of different port areas by gas at orifice pressure of 4 inches of water with differently designed injecting tubes (Values taken from Fig. 17)*

throat diameter. If the area of the one-half-inch throat is plotted as the best injector for the 0.45 square inch port area, the area of the five-eighths-inch throat as the best injector for the 0.70

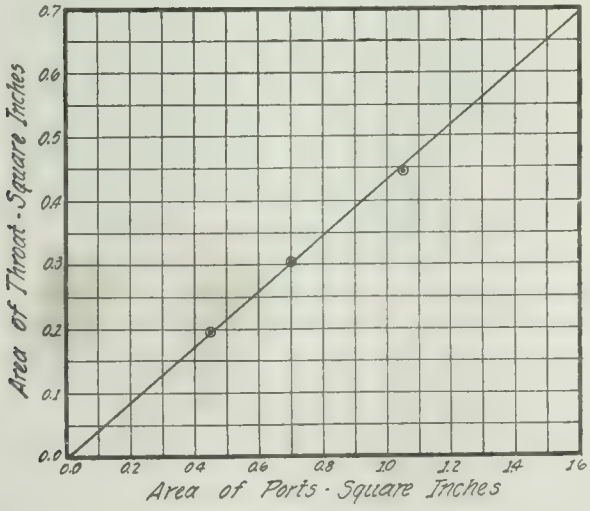


FIG. 19.—*Relation of area of ports to area of throat of injecting tube of good design (Values selected from Fig. 18)*

square inch port area, and the area of the three-fourths-inch throat as the best injector for the 1.05 square inch port area, the points will plot the straight line curve shown in Fig. 19. From this

straight line it is observed that the ratio of throat area of injector to port area of burner should be about 0.43. As a result of the study of our numerous data we believe this relation will hold good over a wide range.

VII. THE BURNER TUBE AND BURNER PORTS

1. CHARACTERISTICS OF A SATISFACTORY BURNER

There are a few appliances in which the character of the flame is of comparatively little importance provided the gas is completely burned. An example of such an appliance would be the ordinary space heater, or the type of warm-air furnace where the products of combustion mix with the air and are delivered into the room. In these appliances the luminous flame would be as efficient as the bunsen flame.

In most cases, however, the temperature of the flame is of great importance and it might be said, in general, that the closer the atmospheric burner approaches the performance of the blast burner the higher will be the efficiency obtained and the wider will be the field of application of such burners.

Different processes will require burners of different characteristics, and to get the very best results one should have a burner designed for the particular quality and composition of gas, and for the pressure available. In domestic appliances this is, of course, impracticable, and the best that can be done is to strike a good average and to make them so that with readjustment they will give fairly satisfactory results over widely varying conditions. In such appliances it is essential that the heat is properly distributed; that the flame is so located as to allow complete combustion without objectional odors and poisonous products; and that high thermal efficiency is not secured by sacrificing other items, such as convenience, simplicity, and safety.

In industrial burners, on the other hand, where large quantities of gas are being used, the cost or inconvenience of changing the design of burners to suit the exact conditions is trifling. The design of industrial burners is worthy of greater attention than has been given the subject.

As a very general statement it can be said that burners should have the following characteristics:

(*a*) For a given size they should have a large rate of consumption. Large rate of consumption means reduction in cost of manufacture, and permits concentration of heat which usually

produces greatest efficiency. Increase in velocity of mixture going through the burner means less heating of the burner.

(*b*) The burner should be capable of operating with a high air-gas ratio, since an air-gas ratio that approaches a theoretical mixture produces a small flame of high temperature.

(*c*) The flame should be of uniform height in all parts of the burner, so that the distribution of heat will be uniform.

(*d*) The burner must stand a considerable variation in the gas pressure, or gas rate, without giving trouble.

(*e*) The flame must not flash back into the burner.

(*f*) The flame must not blow off.

The relative importance of the different characteristics will depend upon the nature of the process and is somewhat a matter of individual judgment.

To make the tests described in this paper as practical as possible it was first thought best to determine the results that could be obtained with burners just as they were supplied by the manufacturer, and then study the effect of improvements on those burners. We soon observed that unless the gas stream was directed exactly into the center of the mixing tube, the results were uncertain and variable. The following precautions are necessary:

1. The shoulder on the air mixer, as well as the end of the burner, must be machined to obtain correct alignment.

2. The threads of the orifice and those of the air mixer holding the orifice must be true to give proper alignment.

3. The orifice must be drilled with extreme care.

With the more constricted throat formed by the venturi tube, the proper alignment of the orifice is absolutely essential if it is desired to obtain the maximum air injection.

2. LIMITING VALUES FOR VELOCITY OF EFFLUX OF DIFFERENT AIR-GAS MIXTURES

The velocity of flame propagation increases rapidly with increase in the air-gas ratio, but decreases again before the theoretical mixture has been reached. The increase in speed of combustion is accompanied by a decrease in the size of the flame and an increase in the temperature of the flame. Fig. 20 shows roughly the limits, for water gas of about 590 Btu, within which one must operate ordinary pipe burners to secure satisfactory conditions.

As shown by the curves, there are limiting values for the maximum and the minimum rates of gas consumption, and these limits vary with the air-gas ratio. For instance, with a 1.35 square inch port area burner, when the velocity of the mixture through the ports exceeds 15 feet per second with a 3 to 1 ratio, the flames are blown from the ports; when the velocity is about 3.5 feet per second the mixture flashes back into the burner. With a higher air-gas ratio the limits of flash back and blow from ports become closer together. It follows, then, that the port area must not be

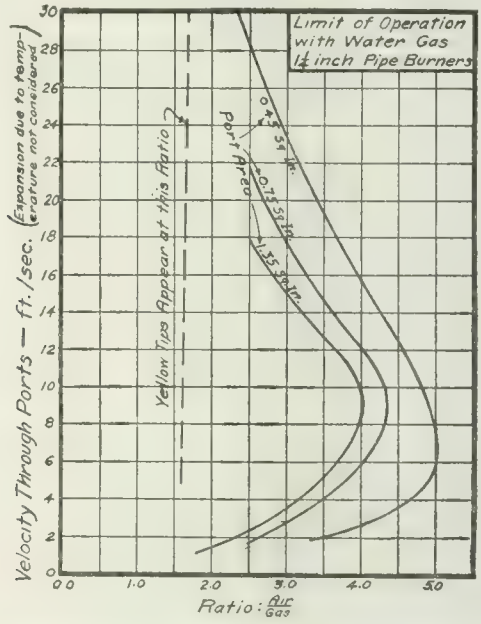


FIG. 20.—*Curves showing velocity of the air-gas mixture through ports of burners when the air-gas ratio is sufficient to cause the flames to blow from the ports*

so large that the velocity of flame propagation exceeds the velocity of the mixture through the ports, nor should the velocity of the outflowing mixture be so great that the flames will blow from the ports.

If a small size port is used it is possible to reduce the velocity of outflowing mixture to a much lower rate than with the larger size ports before a flash back occurs. This is perhaps due to the greater cooling effect of the secondary air. With special burners having small ports, the velocity of outflowing mixture may be reduced very low without a flash back.

3. WHY IT IS EASIER TO CHANGE FROM WATER GAS TO COAL GAS THAN FROM COAL GAS TO WATER GAS

Some discussion of this matter of changing from one gas to another occurred at one of the meetings during the 1919 convention of the American Gas Association. A statement was made that by making water gas of very high heating value it had been found possible to change from coal gas to water gas without adjustment of appliances. Several others related experiences when changes were made from gas of one composition to gas of another composition. No one explained under what conditions these changes were possible. Because it appears that the reason is not clear to everyone and as it is a rather important question, some further explanation may be worth while, although a careful consideration of the preceding sections should have explained the reason.

Most processes, from the ordinary domestic cooking to the large industrial processes, require approximately a certain number of Btu per hour. If the appliances are adjusted to the condition that gives the best results for a light gas, and if a change is made to a heavier gas, such as changing from coal gas to water gas, a less number of cubic feet of gas per hour and a less number of Btu per hour are delivered if the heating values of the two gases are the same. Even though it may be possible to operate the appliance without readjustments the value of the service is decreased and dissatisfaction results. Changing from the lighter to the heavier gas will also increase the air entrainment, as described in previous sections, and the velocity of flame propagation will be greater. The result will be that in most cases the velocity of efflux through the ports will not be sufficient to prevent the mixture from flashing back.

In the case cited above, where the water gas was made of extremely high heating value, the increase in Btu per cubic foot offset, to some extent, the decrease in volume of gas entering the burner, and the increase in illuminants decreased the velocity of burning sufficiently to prevent the flash back. If, when operating with coal gas, the appliances were adjusted for a very soft flame—that is, a low air-gas ratio—less difficulty would be experienced on changing over to water gas.

In changing from the heavier to the lighter gas less difficulty would ordinarily be experienced. With the same pressure the orifice will pass a greater volume of gas to the burner, and the

consumer will merely turn off the excess and rarely have any great trouble, unless the air injection has been so reduced at the same time that the combustion is not complete and the gas burns with a luminous flame which may blacken the utensils.

4. RELATION BETWEEN THE TOTAL PORT AREA AND THE RATE OF CONSUMPTION OF BURNERS

(*a*) BURNERS WITHOUT INJECTING TUBES

With pipe burners as ordinarily constructed without injecting tubes, the necessity of keeping a correct ratio between the port area and the cross-sectional area of the pipe becomes extremely important when it is desired to secure the maximum rate of consumption from the burner and a good injection of primary air. This is well illustrated in the following table, in which the values are taken from Figs. 21, 22, and 23, for burners operated "cold" with gas of 1.0 specific gravity:

TABLE 4.—Rate of Consumption of Pipe Burners Without Injecting Tube

[Air-gas ratio, 3 to 1; specific gravity of gas, 1.0; burner operated "cold"]

Total port area in square inches	Ratio of port area to area of pipe	Cubic feet per hour at 4.0 inches water pressure	Cubic feet per hour per square inch of port area
0.45	0.3	24	53
.75	.5	57	76
1.05	.7	84	80

[Rates of consumption for a 0.65 specific gravity gas calculated from the foregoing values]

0.45	0.3	21.3	47.3
.75	.5	50.6	67.4
1.05	.7	74.0	70.5

From the above table it is seen that, for a burner without an injector, the rate of consumption per square inch of port area increases until the port area is about the cross-sectional area of the pipe. These values are shown later in Table 7, and give the relative rates of consumption of burners with and without injecting tubes. See also Fig. 28 in which all of these values are plotted.

Due to the heating of the burner and the expansion of the mixture within the burner, the rate of consumption is reduced somewhat upon the burner being lighted, and is further reduced when

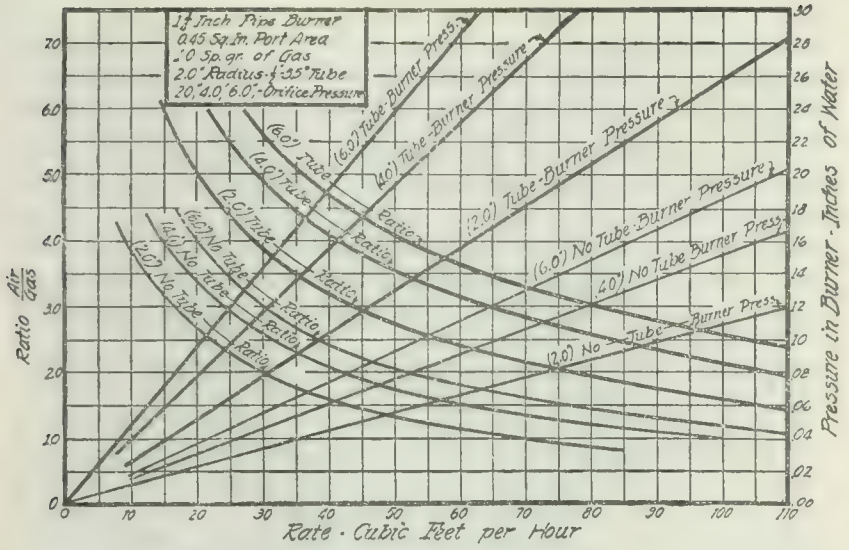


FIG. 21.—*Air injection secured with 1.0 specific gravity gas at orifice pressures of 2, 4, and 6 inches of water in a 0.45 square inch port area burner with and without improved injecting tube*

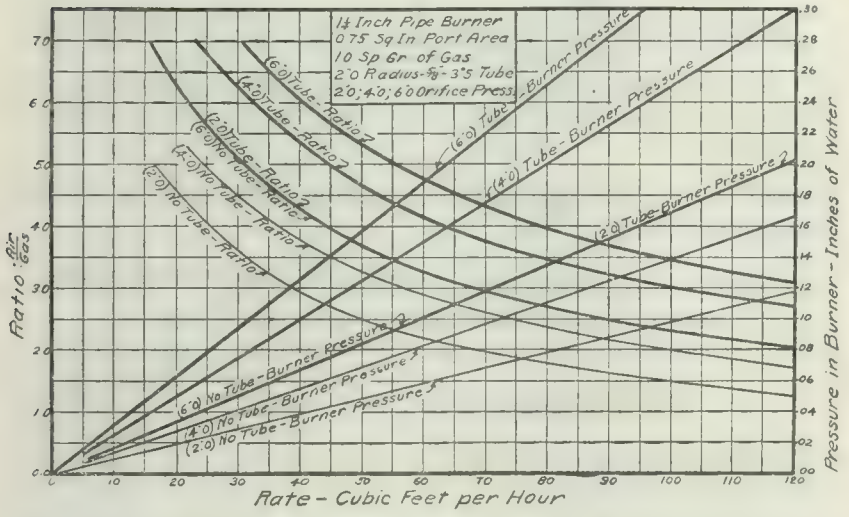


FIG. 22.—*Air injection secured with 1.0 specific gravity gas at orifice pressures of 2, 4, and 6 inches of water in a 0.75 square inch port area burner with and without improved injecting tube*

an object is placed over the burner which will cause some of the heat to be reflected back upon the burner.

Unless the average temperature of the burner is known it is impossible to calculate the reduction in rate of consumption. Just what the heating effect will be is indefinite and will depend upon the installation, but with a little experience one should be able to make the proper allowance for the temperature effect for each condition.

The following table, with values taken from Figs. 24, 25, 26, and 27, shows the relative rates of consumption of these same burners when the burner is lighted and the temperature has

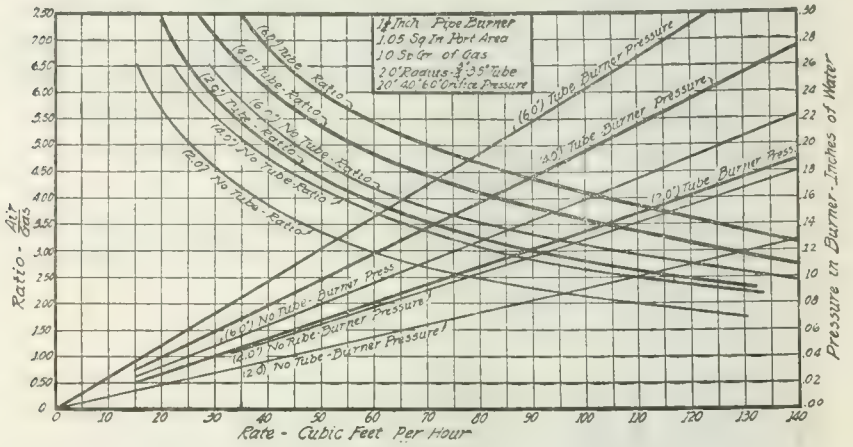

1/4 Inch Pipe Burner
1.05 Sq In Port Area
1.0 Sp Gr of Gas
2.0" Radius-3/4" 35° Tube
30° 40° 60° Orifice Pressure
(6.0") Tube Ratio
(4.0") Tube Ratio
(2.0") Tube Ratio
(6.0") No Tube Ratio
(4.0") No Tube Ratio
(2.0") No Tube Ratio
(6.0") Tube Burner Pressure
(4.0") Tube Burner Pressure
(2.0") Tube Burner Press.
(6.0") No Tube Burner Press.
(4.0") No Tube Burner Pressure
(2.0") No Tube Burner Pressure
Ratio - Air / Gas
Pressure in Burner - Inches of Water
Rate - Cubic Feet Per Hour

FIG. 23.—*Air injection secured with 1.0 specific gravity gas at orifice pressures of 2, 4, and 6 inches of water in a 1.05 square inch port area burner with and without improved injecting tube*

reached the normal that it would assume when burning without an object placed over it:

TABLE 5.—Rate of Consumption of Pipe Burners Without Injecting Tube

[Air-gas ratio, 3 to 1; specific gravity of gas, 0.65; burner at normal temperature]

Total port area in square inches	Ratio of port area to area of pipe	Cubic feet per hour at 4.0 inches water pressure	Cubic feet per hour per square inch of port area
0.45	0.3	18.0	40.0
.75	.5	41.5	55.3
1.05	.7	66.5	63.3
1.35	.9	99.0	73.2

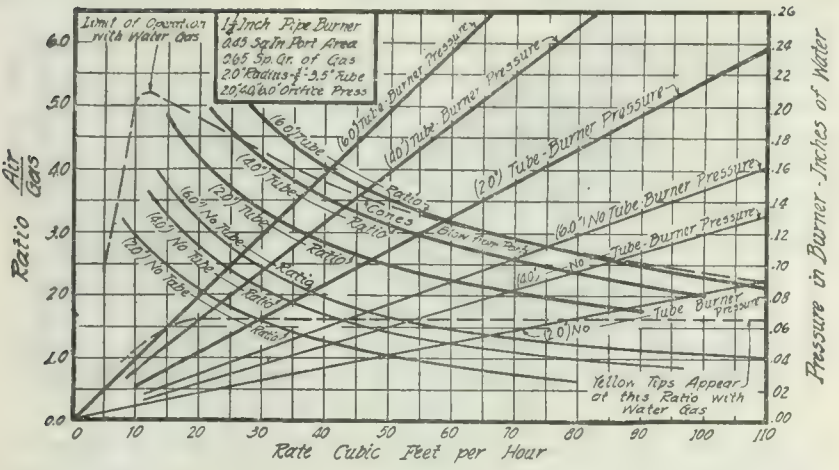


FIG. 24.—*Air injection secured with 0.65 specific gravity gas at orifice perssures of 2, 4, and 6 inches of water in a 0.45 square inch port area burner with and without improved injecting tube*

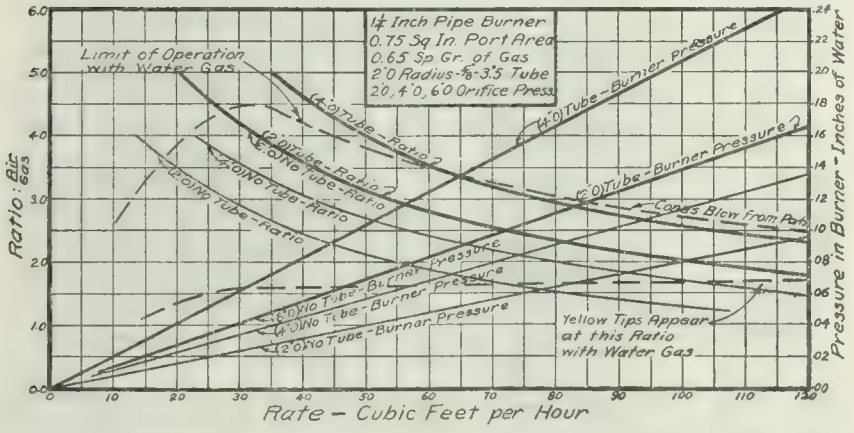

1¼ Inch Pipe Burner
0.75 Sq In. Port Area
0.65 Sp Gr. of Gas
2'0 Radius-⅛-3.5 Tube
20, 40, 60 Orifice Press
Limit of Operation with Water Gas
(4'0) Tube-Burner Pressure
(2'0) Tube-Burner Pressure
(4'0) Tube-Ratio
(2'0) Tube-Ratio
(4'0) No Tube-Ratio
(2'0) No Tube-Ratio
(6'0) No Tube-Burner Pressure
(4'0) No Tube-Burner Pressure
(2'0) No Tube-Burner Pressure
Cones Blow from Ports
Yellow Tips Appear at this Ratio with Water Gas
Ratio: Air/Gas
Pressure in Burner - Inches of Water
Rate - Cubic Feet per Hour

FIG. 25.—*Air injection secured with 0.65 specific gravity gas at orifice pressures of 2, 4, and 6 inches of water in a 0.75 square inch port area burner with and without improved injecting tube*

(*b*) BURNERS WITH INJECTING TUBES

With injecting tubes of good design the curves as shown in Figs. 21 to 27, inclusive, were obtained. These curves show the

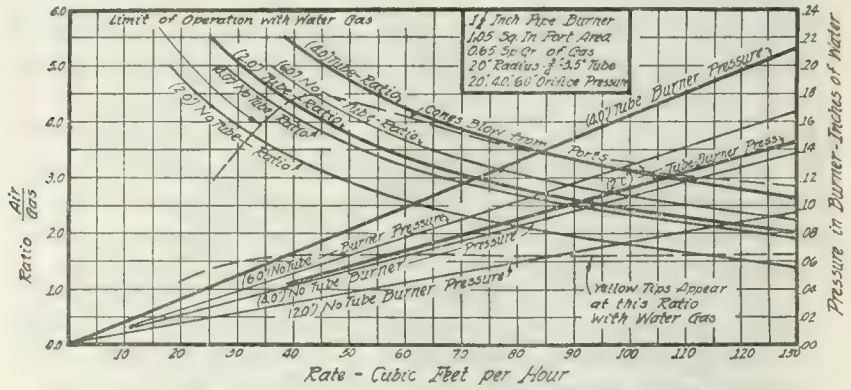

Limit of Operation with Water Gas
1¼ Inch Pipe Burner
1.05 Sq. In. Port Area
0.65 Sp. Gr. of Gas
20" Radius of 3.5" Tube
20" 40" 60" Orifice Pressure
(40") Tube Ratio
(60") No Tube Ratio
(40") No Tube Ratio
(20") No Tube Ratio
Comes Blow from Ports
(40") Tube Burner Pressure
(20") Tube Burner Pressure
Burner Pressure
(60") No Tube Burner Pressure
(40") No Tube Burner Pressure
(20") No Tube Burner Pressure
Yellow Tips Appear at this Ratio with Water Gas
Air / Gas Ratio
Pressure in Burner-Inches of Water
Rate - Cubic Feet per Hour

FIG. 26.—*Air injection secured with 0.65 specific gravity gas at orifice pressures of 2, 4, and 6 inches of water in a 1.05 square inch port area burner with and without improved injecting tube*

rates of consumption of burners of 0.45, 0.75, 1.05, and 1.35 square inches port area.

A still more efficient injecting tube, shown by No. 1 in Fig. 13, was designed later. With this design it is possible to secure

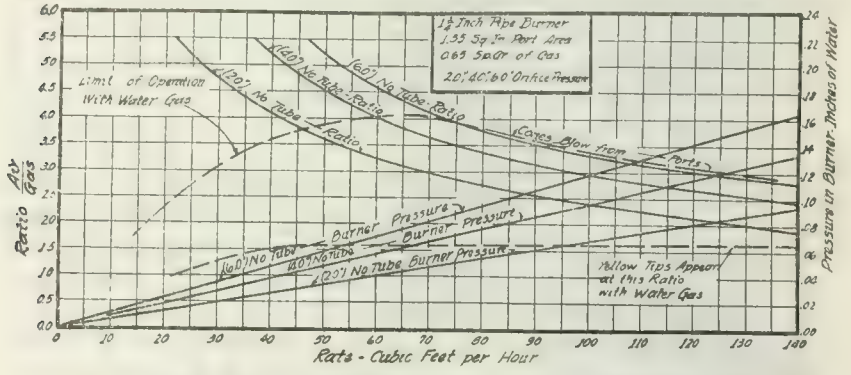

1½ Inch Pipe Burner
1.35 Sq. In. Port Area
0.65 Sp. Gr. of Gas
20" 40" 60" Orifice Pressure
Limit of Operation With Water Gas
(60") No Tube Ratio
(40") No Tube Ratio
(20") No Tube Ratio
Comes Blow from Ports
Burner Pressure
(60") No Tube Burner Pressure
(40") No Tube Burner Pressure
(20") No Tube Burner Pressure
Yellow Tips Appear at this Ratio with Water Gas
Air / Gas Ratio
Pressure in Burner-Inches of Water
Rate - Cubic Feet per Hour

FIG. 27.—*Air injection secured with 0.65 specific gravity gas at orifice pressures of 2, 4, and 6 inches of water in a 1.35 square inch port area burner without improved injecting tube*

slightly higher rates of consumption as indicated from the pressure curves in Fig. 15.

The following table will show the results from the above-mentioned figures for gases of 1.0 and 0.65 specific gravities with burner "cold," and for a 0.65 specific gravity with burner "hot":

TABLE 6.—Rate of Consumption of Pipe Burners With Injecting Tubes

[Air-gas ratio, 3 to 1; specific gravity of gas, 1.0; burners cold]

Total port area in square inches	Ratio of port area to area of pipe	Cubic feet per hour at 4.0 inches water pressure	Cubic feet per hour per square inch of port area
0.45	0.3	65	144.4
.75	.5	101	135.0
1.05	.7	126	120.0
[Values calculated from the foregoing figures, for a gas of 0.65 specific gravity; burners cold]			
0.45	0.3	57.0	126.5
.75	.5	89.0	118.5
1.05	.7	114.0	108.5
[Values for the same burners with gas of 0.65 specific gravity; gas burning]			
0.45	0.3	52.5	116.7
.75	.5	77.5	103.3
1.05	.7	107.5	102.4
1.35	.9	124.0	91.9

Some of the values from the preceding tables have been arranged in the following table, No. 7, to show the relative rates of consumption of burners "cold" and burners "hot," and also to show the percentage of increase in rate of consumption secured with a well-designed injecting tube.

TABLE 7.—Rate of Consumption of Pipe Burners With and Without Injecting Tubes

[Air-gas ratio, 3 to 1; specific gravity of gas, 0.65; values taken from figs. 21 to 27, inclusive; burners cold

Total port area in square inches	Ratio of port area to area of pipe	Cubic feet per hour		Cubic feet per hour per square inch of port area		Per cent increase in capacity
		No injector	Injector	No injector	Injector	
0.45	0.3	21.3	57.0	47.3	126.5	167
.75	.5	50.6	89.0	67.4	118.5	76
1.05	.7	74.0	114.0	70.5	108.5	54
[Rate of consumption with burners lighted]						
0.45	0.3	18.0	52.5	40.0	116.7	192
.75	.5	41.5	77.5	55.3	103.3	87
1.05	.7	66.5	107.5	63.3	102.4	62
1.35	.9	99.0	124.0	73.2	91.9	25

The values from the last table have been plotted in the form of curves in Fig. 28.

These tables and curves show for the "no injecting tube" study that the rate of consumption per square inch of port area increases as the area of ports is made nearer the cross-sectional area of the pipe. They also show that there is a drop in rate of consumption when the burner is lighted.

With the injecting tubes curves were obtained which require some explanation. In this case 1¼-inch pipe burners of different

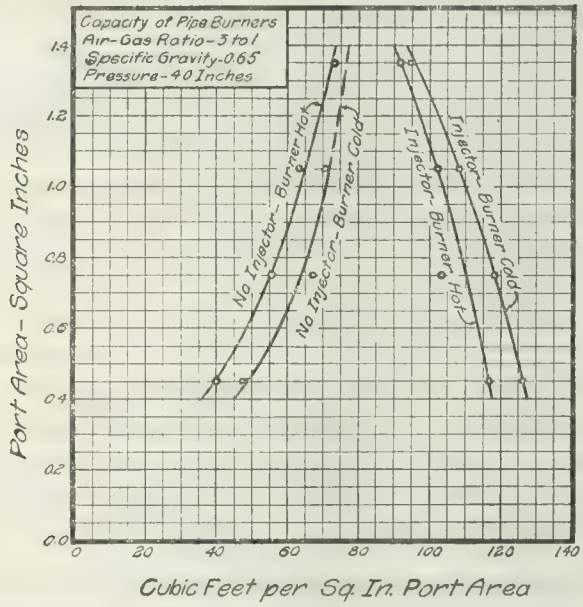


FIG. 28.—*Maximum rate of consumption of 1¼-inch pipe burners with and without improved injecting tubes when using 0.65 specific gravity gas at an air-gas ratio of 3 to 1*

port areas were used. With each change in port area an injector was used that had an area of throat which our previous experiments had shown to be the best for that particular port area. The inlet and outlet angles of all the injectors were the same, but, since the injector with the smallest throat was the longest one, and the cross-sectional area of the pipe was largest, relative to the area of the throat, it allowed the velocity of the mixture to be reduced more gradually and the average static pressure in the burner was greater.

5. EFFECT OF LENGTH OF BURNER ON RATE OF CONSUMPTION

The 0.45 square inch port area burner was less than half as long as the 1.05 square inch port area burner, and it was at first thought that the greater rate of consumption per unit port area secured with the throat was due to increased friction loss with the longer burner. A 1.05 square inch port area burner was made with a double row of ports, and one twice as long with a single row of ports, and it was found that under the same conditions the difference in pressure in the burner was about 1 per cent, thus proving rather conclusively what has already been said in our discussion of Table 7, about the effectiveness of the small throat in a large pipe.

Where the burner is unusually long, or has cross arms, or sharp bends, it is difficult to calculate the friction loss and reduction in rate of consumption, but some allowance will have to be made, depending upon circumstances.

6. EFFECT OF PORT SIZE ON RATE OF CONSUMPTION

With burners having the same port area, but with the size of ports varying from No. 27 to No. 40 drill, no appreciable difference in rate of consumption was found; but with the smaller size port it was possible to turn the gas lower without causing a flash back. This is perhaps due to the secondary air having a greater cooling effect on the small flames, thus reducing the velocity of flame propagation.

7. PRESSURES IN PIPE BURNERS—FIRST AND LAST PORTS

In pipe burners all of the mixture must pass the first port. The velocity of the mixture, therefore, is greatest at this point, while at the last port the velocity is nil. At the first port there are both velocity and static pressures, but at the last port there is, of course, no velocity pressure. The static pressure is the maximum, then, at the last port. The volume of the mixture which issues from a port is dependent upon the static pressure at that port. If the ratio of the port area to the cross-sectional area of the pipe is large, there is a wide difference in static pressures and, consequently, in the volumes of the mixture which issues from the first and last ports. Table 8 contrasts the results of tests made with two ratios of port area to cross-sectional area of pipe.

TABLE 8.—Static Pressures in Pipe Burners—First, Middle, and Last Ports

[Percentage on basis of pressure at last port]

Port area in square inches	Ratio of port area to cross-sectional area	First port	Middle port	Last port
		Per cent	Per cent	Per cent
0.75	0.5	90.0	97.5	100.0
1..35	.9	77.0	92.5	100.0

8. CONE HEIGHTS IN PIPE BURNERS—FIRST AND LAST PORTS

For a given air-gas ratio there is a direct relation between cone height and flame height, pressure in the burner, and the size of the port. The tests shown in the following table indicate the manner in which the heat is distributed along pipe burners.

TABLE 9.—Cone Height Study With Water Gas

[1¼ inch pipe burner; specific gravity, 0.65; port area, 1.35 square inches; size of ports, No. 30 drill; air-gas ratio, 3 to 1]

Orifice pressure, inches of water	Gas rate, cubic feet per hour	First port		Last port	
		Velocity of mixture in burner, feet per second	Cone height in inches	Velocity of mixture in burner, feet per second	Cone height in inches
2	70.5	7.52	0.55	0	0.61
4	100.0	10.70	.63	0	.80
6	122.5	13.10	.67	0	.95

For the particular conditions stated in Table 9 it is seen that the cone height at the first port is: At 2 inches pressure, 90 per cent; at 4 inches pressure, 79 per cent; and at 6 inches pressure, 70 per cent of the cone height at the last port. This difference in cone height is caused by the difference in static pressure in the burner between the first and last ports.

For this condition the ratio of the port area to the cross-sectional area of the pipe was 0.90. At the same orifice pressures, with an air-gas ratio of 3 to 1, any pipe burner with the same ratio of port area to cross-sectional area should give the same proportional differences in cone heights. Since the air-gas mixture leaving each port is the same, it follows that, when there is a difference in cone heights, more heat is liberated at the last port than at the first. It is evident that, if an even distribution

of heat is desired, the burner must be so designed that the difference in velocity of mixture between the first and last ports is not too great.

9. HOW TO CHOOSE A BURNER FOR ANY CONDITION

Much of the foregoing information on the effect of port area and burner size is valuable and is essential for a real understanding of the problem, but it will not serve the everyday requirements of the industrial fuel engineer and appliance man unless the information can be tabulated clearly so that he can select quickly and accurately the proper burner for any given condition.

In view of this fact there has been compiled a series of tables, based on the preceding experimental work and calculations, which show the rate of consumption of various sizes of burners for different pressures and air-gas ratios.

These tables are adapted only to the conditions specified, but if these tables or further modifications of them are found to work in everyday practice, it will be a simple matter to work up such tables for any other given condition.

Since the rates of consumption of burners without injecting tubes increase with increasing port area, it will be difficult to make up tables that will be generally applicable for the various types of burners.

Many of the industrial pipe burners are made at present with a port area within 5 to 10 per cent of the cross-sectional area of the pipe, and this works well in practice for the usual low rate of consumption secured without injecting tubes, and where it is not necessary to have a very uniform distribution of heat. A port area of 20 per cent, or even 40 per cent, less than the cross-sectional area of the pipe might prove to be more satisfactory in many installations.

With an injecting tube the rate of consumption of a burner is increased so much that it is necessary to have the pipe larger in proportion to the port area, otherwise the velocity past the first ports is too great to give good results.

From the curves in Fig. 28, we have taken the value corresponding to the 1.2 square inch port area as a basis for our tables. The value is 70 cubic feet per square inch of port area and may be used with but small error for burners from 0.7 to 0.9 ratio of port area to cross-sectional area. No tables are shown for burners with

injecting tubes because the injectors will vary considerably in practice, and it will be necessary to make up the tables after testing the injector.

In designing an installation the first thing to consider is the volume of gas that will be required for the particular operation. The next point to consider is the nature of the operation, since if the appliance is a drying oven or any other installation requiring hot air, it will be sufficient if the gas is completely burned, in which case the characteristics of the flame are not so important. If, on the other hand, a hot flame is required, and especially where a large quantity of heat is required in a small space, it is absolutely essential that much of the air required for combustion is drawn in as primary air. Increasing the primary air, however, makes it necessary to have a more definite relation between the gas rate and the port area, for, as shown in Fig. 20, a small change in rate will either cause the burner to flash back or the flame to blow from ports.

Having decided on the gas rate and the character of the flame that is required, it will be necessary to know what gas pressure is available during the periods of minimum pressure.

Knowing the gas pressure, the volume of gas required, and the flame characteristics required, it is possible to select from such tables as the following the correct burner for any condition:

TABLE 10.—Rate of Consumption[a] of Pipe Burners Without Injecting Tubes

[Specific gravity of gas, 0.65; area of ports, 0.8 of the cross-sectional area of the pipe]

Nominal size of pipe in inches	Air-gas ratio	Rate of consumption in cubic feet per hour at gas pressures of—				
		1 inch	2 inches	3 inches	4 inches	5 inches
3/4	2.5 to 1	20	28	34	40	44
1	2.5 to 1	32	45	55	64	71
1 1/4	2.5 to 1	55	78	96	111	124
1 1/2	2.5 to 1	76	107	131	151	169
2	2.5 to 1	124	176	216	249	278
3/4	3 to 1	15	21	26	30	33
1	3 to 1	24	34	42	48	54
1 1/4	3 to 1	42	59	73	84	94
1 1/2	3 to 1	57	81	99	114	128
2	3 to 1	94	133	163	188	210

[a] Calculated on the basis that the rate of consumption per square inch of port area is 70 cubic feet per hour at 3 to 1 ratio

Where it is desired to increase or decrease the ratio of the port area to the cross-sectional area of the pipe, it is necessary to increase or decrease the values in the same ratio as the port area is changed. This applies for ratios from 0.7 to 0.9. For other changes in the ratio see curves giving rate of consumption per square inch of port area for different burners.

TABLE 11.—Rate of Consumption [a] of Pipe Burners Without Injecting Tube

[Specific gravity of gas, 0.35; area of ports, 0.8 of the cross-sectional area of the pipe]

Nominal size of pipe in inches	Air-gas ratio	Rate of consumption in cubic feet per hour at gas pressures of—				
		1 inch	2 inches	3 inches	4 inches	5 inches
3/4	2.5 to 1	16	23	28	32	36
1	2.5 to 1	25	36	44	51	57
1 1/4	2.5 to 1	45	64	78	90	101
1 1/2	2.5 to 1	61	86	106	122	137
2	2.5 to 1	101	143	175	202	226
3/4	3 to 1	12	17	21	24	27
1	3 to 1	19	27	33	39	43
1 1/4	3 to 1	33	47	58	67	75
1 1/2	3 to 1	46	65	78	91	102
2	3 to 1	75	106	130	150	168

[a] Calculated on the basis that the rate of consumption per square inch of port area is 50 cubic feet per hour at 3 to 1 ratio.

Where it is desired to increase or decrease the ratio of the port area to the cross-sectional area of the pipe, it is necessary to increase or decrease the values in the same ratio as the port area is changed. This applies for ratios from 0.7 to 0.9. For other changes in the ratio see curves giving rate of consumption per square inch of port area for different burners.

VIII. THE AIR SHUTTER

Domestic appliances are generally operated at different pressures in different localities and must be so designed by the manufacturer that they will give satisfactory service even with extremely low pressures. Such burners, when operated with medium and high pressures, inject too large a volume of primary air and an adjustment of the air shutter is necessary.

Most municipal ordinances require that the minimum gas pressure shall not be less than 2 inches, and it would seem that in designing a burner for the average condition pressures lower than 2 inches should not be considered. If this is conceded, it can be said that good design in a domestic range burner demands that, when the burner is operated with artificial gas at its maximum rate of consumption—from 15 to 18 cubic feet per hour at 2 inches pressure—the burner should operate satisfactorily and give good flame characteristics with the air shutter wide open. If this is not possible, the port area is not correct for the volume of gas, and what has been gained in injecting power through the use of good orifices, injecting tubes, etc., has been lost by restricting the free flow of air into the injector. This is equally true for industrial appliances, but as these are usually designed more nearly for the existing conditions, it is not necessary to make so much allowance in design for varying conditions as in the case of domestic burners.

1. AREA OF AIR-SHUTTER OPENING REQUIRED

The maximum rate at which a burner is to be operated must be known. It is also necessary to know what the minimum size of air-shutter opening should be in order to get the required volume of air into the burner. To form some opinion of the area of air-shutter opening required, a burner was operated with the air shutter in a position where it offered no resistance to the flow of air, and the total volume of air injected was determined. By

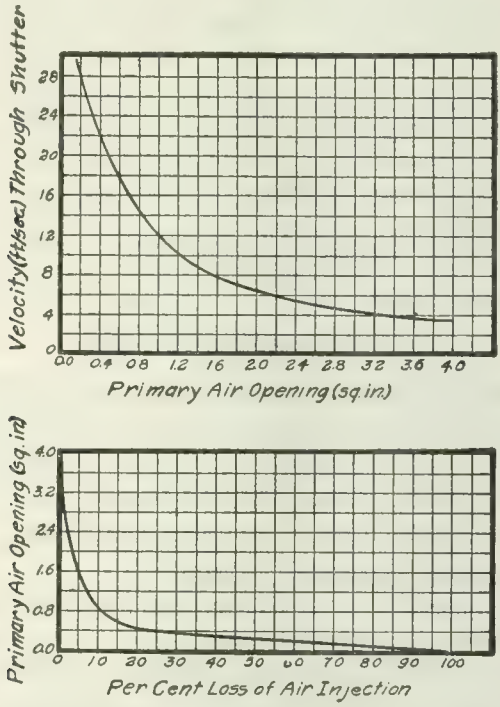


FIG. 29.—*Velocity of air through shutter with different areas of shutter opening and relation of air injection to velocity of air through shutter*

gradually closing the air inlet and observing the effect of the air injection the values shown in Table 12 were obtained.

These values have been plotted in the curves of Fig. 29, and show the loss of injecting power with the decrease of the area of air inlet for the particular conditions stated in Table 12. No attempt to draw any definite conclusions from these curves is made, since there are a number of things to be taken into consideration. The air injection will vary with the momentum of the gas stream, the size of the burner, the design of the injector, and the area of the

air-shutter opening. In general, one might say that to keep the loss of air injection down to 1 or 2 per cent, it is necessary to have the shutter opening large enough that the velocity of the air through the opening does not exceed 4 or 5 feet per second.

TABLE 12.—Reduction in the Volume of Air Injected When Closing the Air Shutter

Gas rate, 77.0 cubic feet per hour; specific gravity of gas, 1.0; gas pressure, 4 inches; burner, 1.25-inch diameter, with 1.05 square inch port area, with injector]

Total air injected (cubic feet)	Area of air inlet (square inches)	Velocity through air inlet (feet per second)
318	4.0	3.18
317	3.2	3.91
313	2.0	6.26
305	1.4	8.72
285	.8	14.25
260	.6	17.35
210	.4	19.13
138	.2	27.56

IX. SUMMARY

The investigation of the design of the atmospheric burner has resulted in the development of improved methods for studying burner operation. With the apparatus that has been developed one can determine quickly and accurately, for any burner, under any condition of operation, the volume of air injected. The limits of operation for any burner with any quality of gas can be determined. From these limits one can draw fairly definite conclusions regarding the design of burners for any predetermined condition of operation. In order to thoroughly investigate burner design it was found necessary to study the burner orifice and the theory of flow of gas through different types of orifices, the principles governing air injection, the injecting tube, the burner and burner ports, and the air shutter.

1. THE ORIFICE

The orifices have been divided into two types—the channel type and the sharp-edge type. In Fig. 6 orifices Nos. 1, 2, 3, and 4 are designated as sharp-edge type and No. 5 as channel type.

(*a*) SHARP-EDGE ORIFICE—DISCHARGE COEFFICIENT

The discharge coefficient of a sharp-edge orifice with a given angle of approach is a constant for ordinary sizes of gas orifices and over the usual range of gas pressures.

When the discharge coefficient (K) was determined for sharp-edge orifices with different angles of approach it was found that it varies from about 0.605 for a 90° approach to about 0.875 for an 8° approach. The results have been plotted in Fig. 9*b*.

(*b*) SHARP-EDGE ORIFICE—LOSS OF AIR INJECTION

With the sharp-edge types of orifices the loss of air injection was found to be exactly the same for all designs. The results from which the conclusion is drawn are plotted in Fig. 10.

(*c*) CHANNEL ORIFICE—DISCHARGE COEFFICIENT

It was found that with the orifices of the channel type the coefficient will vary not only with a change in the angle of approach but also with a change in the length of channel. In Table 1 and Fig. 7 are shown the results obtained with orifices of different lengths of channel in which the angle of approach was a constant of 45°.

(*d*) CHANNEL ORIFICE—LOSS OF AIR INJECTION

Orifices of the channel type invariably produced a pressure in the burner which was less than that produced by the sharp-edge type when the gas rate of the two types was the same and was produced by the same orifice pressure. The air injection was, therefore, less with the channel type. Fig. 11*b* shows the loss of air injection for different lengths of channel in which the angle of approach was 45°.

(*e*) RATE OF FLOW OF ORIFICE

If the rate of flow with an orifice is desired, or the size of orifice is required for any given gas rate, it is necessary to know the gas pressure, the specific gravity of the gas, and the discharge coefficient of the orifice. The following formulas will enable one to calculate readily any one unknown value by substituting in the formula the known values:

$$Q = 1658.5\ KA\sqrt{\frac{H}{d}} \quad \text{or} \quad A = \frac{Q}{1658.5\ K\sqrt{\frac{d}{H}}}$$

Where Q = rate of flow from orifice in cubic feet per hour;
A = area of orifice in square inches;
K = orifice constant, or discharge coefficient;
H = orifice pressure in inches of water;
d = specific gravity of gas (air = 1.0).

2. PRINCIPLES GOVERNING AIR INJECTION

From the fundamental theory and the relations which have been illustrated by examples it is possible to summarize the most important relations as follows:

1. The ratio between the momentum of the gas stream and the momentum of the stream of the mixture is always the same for a burner of a given design irrespective of orifice pressure, specific gravity of gas, or the volume of the air and gas mixture going through the burner.

2. Where the gas rate is increased by change of pressure, the momenta of the gas streams are directly proportional to the pressures.

3. When the same volumes of gases of different specific gravities issue from different orifices under the same orifice pressure the momenta of the gas streams are proportional to the square roots of the specific gravities of the gases.

4. When the pressure is changed to give the same gas rate for gases of different specific gravity, the air entrainment is proportional to the specific gravities.

5. The pressure at any one point in the burner increases in direct proportion to the increase in the momentum of the stream of the mixture.

If the volume of air injected into a given burner with a gas of a given specific gravity at a given pressure and the gas rate is known, it is possible to calculate from the above-stated relations the volume of air injected with a gas of any other specific gravity, gas rate, or gas pressure.

3. THE INJECTING TUBE

The advantages of having an injector of good design to secure a high injection of primary air has been illustrated by tables and a large number of curves. The general design of an injecting tube that produced the greatest injection of primary air was determined and is shown by injector No. 1, Fig. 13. If it is impracticable to use injectors of this design because of limitations in the size of the burners, it is possible to determine from Table 3 the relative injecting power of other designs.

There is a definite relation between the area of the throat of the injecting tube and the port area of the burner that will give a maximum injection of primary air. It has been shown that the area of the injector throat should be about 43 per cent of the area of the burner ports as shown by Fig. 19.

4. BURNER TUBE AND BURNER PORTS

In the section on the burner tube the characteristics required in a satisfactory burner have been discussed, and we have shown for burners without injecting tubes how the rate of consumption per square inch of port area increases with increase in the port area. The rate of consumption of burners with injecting tubes has been shown in the same way. From the tabulations and curves we have taken the rate of consumption per square inch of port area corresponding to an average burner and have made up Tables 10 and 11 showing the rates of consumption of various sizes of burners for different gas pressures.

5. THE AIR SHUTTER

The velocity of the air through the air opening will depend upon the area of the air opening, the momentum of the gas stream issuing from the orifice, the area of the burner ports, and the design of the injector. The opening in the air shutter must be large enough to allow a free and unrestricted flow of air into the burner. From a few curves that have been shown, it seems that the area of the air opening should be of such size that the velocity of the air through the opening does not exceed 4 or 5 feet per second.

6. CONCLUSION

On account of its simplicity, low cost, and reliability the atmospheric burner is well adapted for domestic and most of the smaller industrial purposes. If it is possible to widen the range within which such burners can be operated efficiently and without adjustments, and design them to meet the needs of any particular purpose, it will make gas fuel much more valuable and will broaden its field of application. With this in view, this Bureau has been conducting experiments upon the efficiency and performance of atmospheric burners, both with natural and artificial gas, and the results will be reported in subsequent papers.

WASHINGTON, December 16, 1920.

DEPARTMENT OF COMMERCE

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BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 194

A PRELIMINARY STUDY OF TEARING INSTRUMENTS AND TEARING TEST METHODS FOR PAPER TESTING

BY

PAUL L. HOUSTON, Associate Physicist

Bureau of Standards

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A PRELIMINARY STUDY OF TEARING INSTRUMENTS AND TEARING TEST METHODS FOR PAPER TESTING

By Paul L. Houston

ABSTRACT

In this technologic paper a study is made of the relative effect of different sizes of test samples on the tearing strength of paper. A great number of samples of commercial papers are torn on three different instruments, using different sizes of test samples and also the same sizes of test samples. Data are collected, accordingly, to show that the larger the test sample the greater are the values of tearing strength. The reason for this is brought out as fabric assistance, which is of considerable importance in the textile industry.

The three instruments used in this study are a tensile-strength instrument and two types of instruments for determining the tearing strength of paper. These two types of instruments are called type I and type II. Type I is a recording instrument, while type II is a nonrecording instrument. A study is then made of these two types of tearing instruments for the purpose of investigating their accuracy and reliability, so that the results of this investigation may benefit the paper industry.

Conclusions are drawn up to show that the type II nonrecording instrument is the more reliable of the two and is within a 5 per cent error on the majority of papers.

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I. INTRODUCTION

1. STATEMENT OF PROBLEM

For some time it has been a recognized fact in the paper industry that there is a need for an instrument that will give numerically the tearing strength of paper. Considerable work has been done along this line at different laboratories. The result has been that two types of tearing instruments have been invented and are used to some extent. One of these types is a recording instrument, giving a curve showing the maximum and minimum tearing strength of paper and the results of five initial tears. The other type gives merely the maximum tearing strength by weighing or measuring the load applied to the paper to tear it. Both types have their good and their bad points, which will be brought to light in the following study.

II. DISCUSSION OF TEST METHODS

1. PRACTICAL TESTS WITH DIFFERENT SIZE TEST SAMPLES

Before taking up a study of the instruments themselves it must be remembered that the tearing strength of paper depends a great deal on the size of the test sample. For instance, a series of bag papers that had previously been tested for weight (in pounds) of the standard size ream 25×40—500 and for bursting strength were torn on the tensile-strength instrument described in Bureau's Circular No. 107, pages 15–17, and on the two types of tearing instruments mentioned above. For convenience, let the recording instrument be called type I and the nonrecording instrument type II. The test samples were cut 1 by 8 inches for the tensile-strength machine, 2¾ by 1⅛ inches for type I instrument, and 4 by 10 inches for type II instrument. All samples were torn the machine direction, beginning at a point just halfway across the width of the test sample, and an average of 10 tests was taken for each result. The results in Table 1 were obtained. It might be well to mention here that all tests in this study were conducted in a constant-temperature and humidity room, where all test samples before they were used were conditioned for two hours at a temperature of 70° F and a relative humidity of 65 per cent. From these data it is to be noted that the larger the test samples are in size the higher are the values of tearing strength.

TABLE 1.—Relative Effect of Different Size Test Samples on Tearing Strength of Bag Paper, Using Tensile Strength, Type I, and Type II Instruments

[Test samples: Tensile strength, 1 by 8 inches; type I, $2^3/_4$ by $1^1/_8$ inches; type II, 4 by 10 inches]

Bag paper identification numbers	Weight of the standard-size ream, 25×40—500	Bursting strength	Tearing strength		
			Tensile strength	Type I	Type II
	Pounds	Points	Grams	Grams	Grams
16 081	32	11	30.8	15.75	33.6
16 082	44	23	58.3	34.80	76.4
16 083	52	18	47.5	38.40	68.4
16 084	49	29	86.6	58.80	96.2
16 085	53	36	91.6	59.80	113.2
16 086	61	36	108.3	63.40	128.6
16 087	32	32	50.0	33.50	64.0
16 088	46	36	100.0	43.20	118.0
16 089	77	54	133.3	90.60	179.2
16 090	77	31	97.5	55.50	149.2
16 091	110	94	270.8	188.00	338.4
16 092	147	125	433.3	256.50	471.4
16 093	39	21	38.3	22.40	60.6
16 094	45	25	43.3	30.80	73.2
16 095	62	33	83.3	62.80	135.6
16 096	45	18	79.1	52.80	94.6
16 097	38	22	101.6	73.10	84.0
16 098	51	33	116.6	74.60	121.6
16 099	65	47	165.6	108.00	179.2
16 100	43	24	78.3	63.50	85.2
16 101	51	28	61.6	39.50	83.0
16 102	72	46	136.6	116.50	186.6
16 103	52	30	73.3	49.90	96.2
16 104	76	41	129.1	88.50	166.8

In order to illustrate this point further, a series of writing papers that had already been tested for weight (in pounds) of the standard size ream 25×40—500 and for bursting strength was torn on the tensile-strength machine and type II nonrecording instrument. Test samples were cut 1 by 8 inches for the tensile-strength apparatus and 4 by 10 inches for the type II instrument, and an average of 10 tests was taken for each result. The data in Table 2 show the same results as in Table 1.

TABLE 2.—Relative Effect of Different Size Test Samples on Tearing Strength of Writing Paper, Using Tensile Strength and Type II Instruments

[Test samples: Tensile strength, 1 by 8 inches; type II, 4 by 10 inches]

Writing paper identification numbers	Kind of writing paper	Weight of standard-size ream 25×40—500	Bursting strength	Tearing strength		Ratio of tensile strength to type II
				Tensile strength	Type II	
		Pounds	Points	Grams	Grams	Per cent
14 731	Bond	86	55	156.9	181.7	86.4
14 732	Writing	46	19	32.4	54.5	59.5
14 733	Copying	24	10	29.0	34.2	84.8
14 734	Printing	55	27	50.2	77.8	64.5
14 735	do	44	19	28.5	44.8	63.6
14 736	Manifold	10	5	13.8	16.8	82.2
14 737	Ledger	98	72	167.2	230.4	72.6

2. PRACTICAL TESTS WITH SAME SIZE TEST SAMPLES

On the other hand, if the test samples are cut the same size, very good check results will be obtained. Note the following results in Table 3 on the above series of long-fibered bag papers when test samples were cut 1 by 8 inches for both tensile-strength and type II instruments. An average of 10 tests was taken for each result.

TABLE 3.—Relative Effect of Same Size Test Samples on Tearing Strength of Bag Paper, Using Tensile Strength and Type II Instruments

[All test samples were 1 by 8 inches]

Bag paper identification numbers	Weight of the standard-size ream 25X 40—500	Bursting strength	Tearing strength		Ratio of tensile strength to Type II
			Tensile strength	Type II	
	Pounds	Points	Grams	Grams	Per cent
16 081	32	11	20.8	21.0	99.0
16 082	44	23	58.3	58.0	100.2
16 083	52	18	47.5	49.6	95.8
16 084	49	29	86.6	90.2	96.1
16 085	53	36	91.6	93.8	97.7
16 086	61	36	108.3	113.0	96.0
16 087	32	22	50.0	50.0	100.0
16 088	46	35	100.0	92.0	108.6
16 089	77	54	133.3	142.2	93.8
16 090	77	31	97.5	104.8	93.0
16 091	110	94	270.8	271.0	99.7
16 092	147	125	433.3	442.8	97.8
16 093	39	21	38.3	41.2	93.0
16 094	45	25	43.3	44.8	96.6
16 095	62	33	83.3	90.4	92.3
16 096	45	28	79.1	74.2	106.6
16 097	38	22	101.6	96.6	102.6
16 098	51	33	116.6	113.4	102.7
16 099	65	47	166.6	160.0	104.0
16 100	43	24	78.3	70.6	110.8
16 101	51	28	61.6	66.8	92.3
16 102	72	46	136.6	139.6	97.8
16 103	52	30	73.3	74.0	99.1
16 104	76	41	129.1	129.6	99.7

3. FABRIC ASSISTANCE

After studying the action of the paper as it was torn on each instrument it became evident that each fiber in the path of the tear received assistance from all the fibers adjoining as far as the edges of the test samples. Also, it was noted that the larger the test sample was the greater was the amount of assistance that the fibers in the path of the tear received from the adjoining fibers, provided, of course, that the beginning of the tear was always the same distance from the end of the test sample and just halfway across the width of the test sample. In the textile industry this effect is called fabric assistance and is of considerable importance.

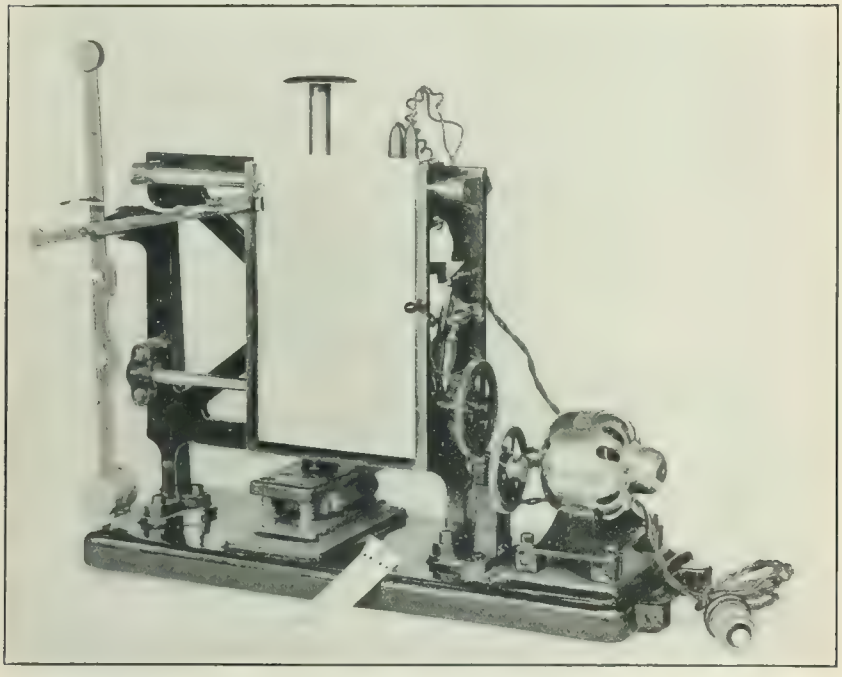

FIG. 1. *The recording instrument, type I*

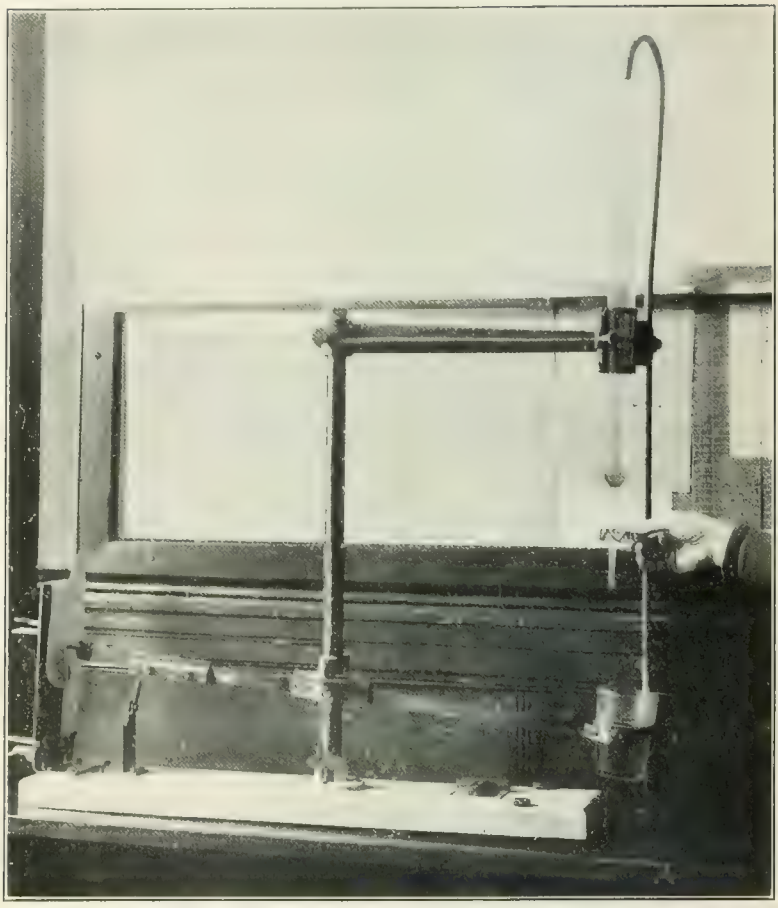

FIG. 3.—*The nonrecording instrument, type II*

III. DISCUSSION OF A RECORDING INSTRUMENT

1. DESCRIPTION OF INSTRUMENT

Since it is found that it is necessary to cut test samples the same size in order to compare the tearing strength of papers, it is now best to proceed to a study of the instruments themselves. The recording instrument, type I, Fig. 1, is the type of instrument preferred for laboratory use. However, such an instrument should be very delicate, sensitive, and accurate. There should be the least possible amount of friction between the chart and pen point. The recording arm should move freely, with as little friction as possible in the bearing. However, there seems to be no instrument of this caliber on the market to-day. The instrument under study has the appearance of a delicate, sensitive, and accurate little machine, but has certain defects which will be discussed later. It is composed of a sliding plate on which the chart rests and a recording arm which moves on a pin-slot bearing and holds a small glass capillary pen with a platinum point. The test sample of paper is cut by means of a die in such a way as to give five initial tears and a curve showing the maximum and minimum tear. Small angular slits in the path of the tear make possible these five initial tears, and a slit down the center of one end of the sample makes it possible for the starting point of the tear to be always the same distance from the edges of the paper. The test sample is placed on the instrument so that one half of the slitted end is fastened to a pin on the sliding plate, while the other half is fastened to a pin on the recording arm. The instrument is motor driven, and as the plate slides to one side the paper is torn and the recording arm is forced down to register on the chart the load in grams necessary to tear the paper. Two different weights may be suspended at three different positions on the projection of the recording arm as factors or multiples of the gram readings on the chart.

2. CALIBRATION TEST

Before operating any instrument of this kind it is always best to calibrate it to determine its accuracy. This was done by hanging dead-weights on the recording arm. With pen point barely touching the chart, there was so much friction in the pin-slot bearing that it was almost impossible to calibrate the instrument at all, as the swinging pen point would stop almost anywhere. However, the following correction curves, Fig. 2, were obtained

for factors, 5, 10, 20, 50, and 100, but they are not reliable for reasons mentioned above. It was impossible to calibrate the instrument for any factor below 5.

3. PERFORMANCE TEST

After attempting to calibrate this instrument it was thought best to tear on it a number of samples of paper, using the different weights to represent factors. The same bag papers were used as before, and from them three identical sets of test samples (numbered 1, 2, and 3) were cut 2¾ inches long and 1⅛ inches wide. Each test sample was torn in the machine direction, and the average of the five initial tears was taken as the tearing strength of the paper. In the following data, Table 4, it is to be noted that different results were obtained for different factors used. The values which are given in this table were directly observed and have not been corrected by use of the calibration curves of Fig. 2.

TABLE 4.—Relative Effect of Different Factors on Tearing Strength of Bag Paper, Using Three Identical Sets of Test Samples and Type I Instrument

[All test samples were 2¾ by 1⅛ inches]

Bag paper identification numbers	Set 1		Set 2		Set 3	
	Factors	Tearing strength	Factors	Tearing strength	Factors	Tearing strength
		Grams		Grams		Grams
16 081	2	15.7	5	17.8	10	18.6
16 082	5	30.0	10	34.8	20	33.6
16 083	5	29.1	10	35.3	20	38.4
16 084	5	41.7	10	58.8	20	53.2
16 085	5	42.3	10	59.8	20	54.2
16 086	10	64.2	20	63.4	50	80.0
16 087	5	33.5	10	24.5	20	30.8
16 088	5	41.6	10	43.2	20	38.8
16 089	20	90.6	50	105.5	100	115.0
16 090	20	54.6	50	55.5	100	72.0
16 091	20	177.2	50	188.0	100	189.0
16 092	50	256.5	100	283.0	.....	.....
16 093	5	22.4	10	30.8	20	22.2
16 094	5	29.6	10	30.8	20	36.7
16 095	20	62.8	50	65.5	100	75.0
16 096	5	43.1	10	52.8	20	46.2
16 097	10	73.1	20	61.0	50	76.0
16 098	10	74.6	20	74.0	50	72.5
16 099	20	108.0	50	114.0	100	152.0
16 100	10	63.5	20	37.2	50	60.8
16 101	10	39.5	20	44.6	50	56.0
16 102	20	101.2	50	116.5	100	146.0
16 103	10	49.9	20	41.2	50	49.0
16 104	20	96.2	50	88.5	100	101.0

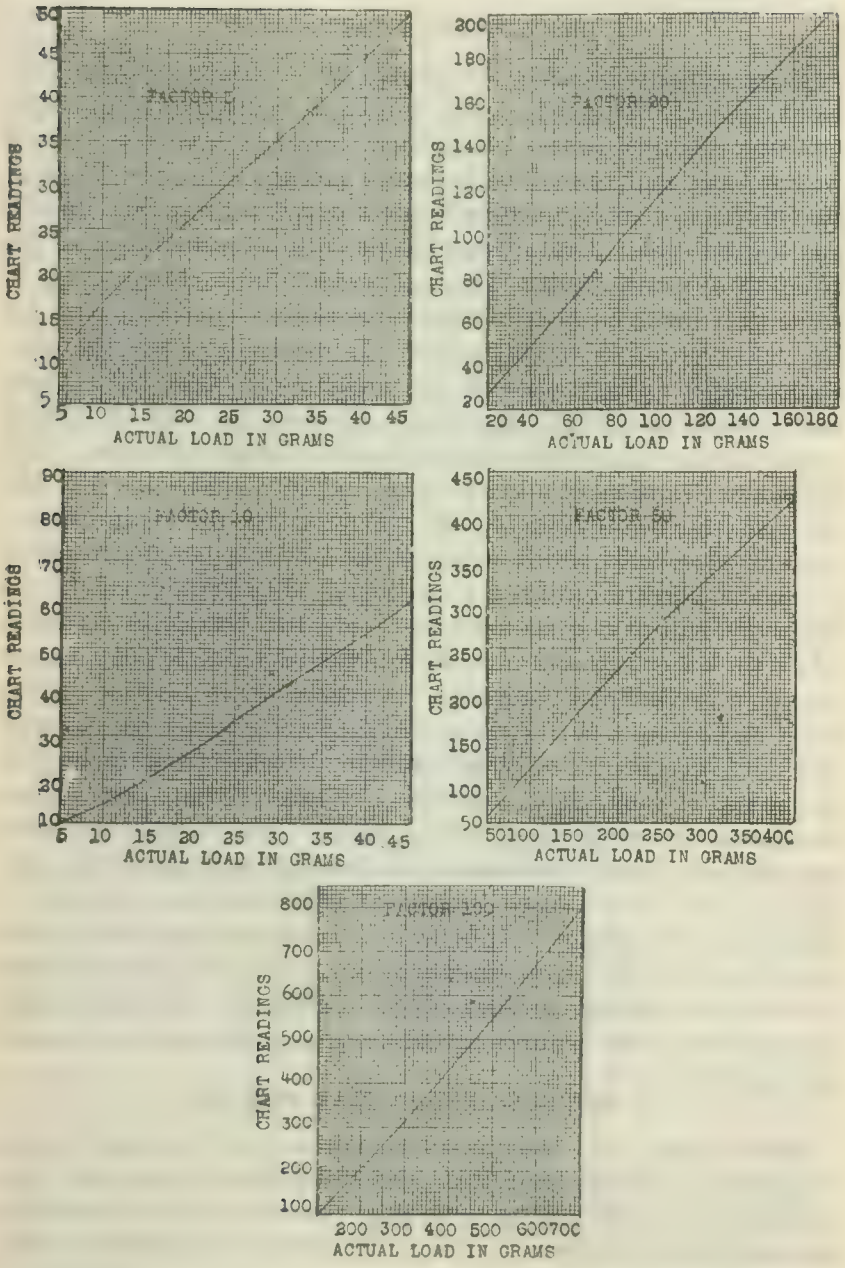

FIG. 2.—*Diagrams showing correction curves for different factors (recording instrument, type I)*

IV. DISCUSSION OF A NONRECORDING INSTRUMENT

1. DESCRIPTION OF INSTRUMENT

There will now be taken up a study of the nonrecording instrument, type II, Fig. 3, which is a simplified instrument adapted for mill use. An instrument for mill use should be accurate, at least within 5 per cent, should be foolproof and yet so simple of operation that paper-mill machine tenders can handle it, and should check itself under standard conditions within the variation of the strength of paper itself. The instrument under study seems to come near being an instrument of this caliber when it is used for bag paper or heavy writing paper. It was built to test the tearing strength of bag paper in the machine direction, since paper bags usually tear in this direction. It could be used on writing paper to determine the tearing strength in either direction. The instrument consists of a beam balancing on a knife-edge (the two arms of the beam being equal). One end of the beam holds a 300-cm^3 glass into which water as the load may be poured from a 500-cm^3 burette until the paper tears. At the other end of the beam one half of one slitted end of the test sample is clamped, while the other half and other end of the test sample are clamped against a vertical plate opposite. A special die is used for cutting a slit and eyelet hole at each end in the middle of the test sample. The cubic centimeter or gram readings are taken from the burette at the end of each tearing operation. These readings indicate the maximum tearing strength of the paper. Two weights may also be used and placed, if necessary, at definite intervals on the glass side of the beam. Each weight at each position represents a certain load in grams. (It is well to state here that due to the facts that the two arms of the beam are equal, and that there is very little friction because of the knife-edge, the force at the tearing end of the beam is actually equal to the weight of water in the glass.)

2. ERROR CAUSED BY BEAM POSITION

Before using this instrument extensively for tearing it was thought best to study it from the standpoint of physics. For instance, it was decided to determine the error due to changes in moment of force caused by changes in position of the beam during the process of tearing. Also, were there errors caused by the force exerted by the falling stream of water? Let there first be made a study of the different moments of force caused by different positions of the beam. Note the following diagram of beam, Fig. 4.

Let BC represent the beam and AR represent a pointer which is fastened at the center A of the beam and at right angles to it, and which determines by its position in respect to the graduations below, numbered 0, 1, and 2, whether the beam is balanced (and in this case horizontal) or is one or two graduations off balance. It is evident that when the beam is one or two graduations off balance the moment of applied force decreases as the number of graduations off balance of the beam increases. In order to find the amount of the error introduced by the changes in moment of force caused by changes in position of the beam, the moment of the applied force was calculated for three different positions of the beam, using 100 g as the weight of water in the glass. Referring to the diagram, Fig. 4, let MF and NG represent the two positions of the beam BC when it is one and two graduations off balance, as indicated by the pointer AR, which at the same time

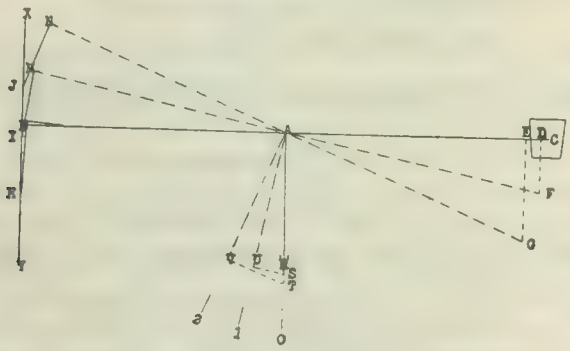

FIG. 4.—*Diagram showing change in effect of weight of water with change in position of beam (nonrecording instrument, type II)*

takes the respective position AU and AV. By drawing the dotted line US perpendicular to AU at point U and dotted line VT perpendicular to AV at point V, and by extending line AR to intersect dotted line US at S and dotted line VT at T, right-angled triangles AUS and AVT are formed. By drawing dotted lines GE and FD from points G and F and perpendicular to AC, right-angled triangles ADF and AEG are formed. Of these triangles angle DAF = angle SAU, and angle EAG = angle TAV. Distances AR, US, VT, and AC have been measured very carefully on the type II apparatus, as follows:

$$AR = AU = AV = 8.9 \text{ cm};$$
$$US = .48 \text{ cm};$$
$$VT = .96 \text{ cm};$$
$$AC = 35.56 \text{ cm}.$$

Referring to triangle AUS,

$$\frac{US}{AU} = \text{tan angle } SAU. \tag{1}$$

Substituting in (1) 0.48 for US and 8.9 for AU, we get:

$$\frac{0.48}{8.9} = \text{tan angle } SAU;$$

$$\text{or, } 0.0539 = \text{tan angle } SAU.$$

Referring to a table of trigonometric functions, it is found that if tan angle $SAU = 0.0539$,

$$\text{cos angle } SAU = 0.9986. \tag{2}$$

Since angle SAU = angle DAF, by substituting in (2) cos angle DAF for cos angle SAU, we get:

$$\text{Cos angle } DAF = 0.9986.$$

Referring to triangle ADF,

$$\text{cos angle } DAF = \frac{AD}{AF}. \tag{3}$$

Substituting in (3) 0.9986 for cos angle DAF, we get:

$$\frac{AD}{AF} = 0.9986;$$

$$\text{or, } AD = AF \times 0.9986. \tag{4}$$

Since $AF = AC = 35.56$, by substituting in (4) 35.56 for AF, we get:

$$AD = 35.56 \times 0.9986;$$

$$\text{or, } AD = 35.51.$$

When the beam is in the position represented by MF and the pointer is at the position of graduation, number 1, the moment of force is represented by the formula:

$$\text{Moment of force} = AD \times \text{weight of water.} \tag{5}$$

Since 100 g were taken as the load in this case, by substituting in (5) 100 for weight of water and 35.51 for AD, we get:

$$\text{Moment of force} = 35.51 \times 100 = 3551.$$

Referring to diagram, Fig. 4, when the beam is in the position represented by NG and the pointer is at the position of graduation, number 2, the moment of force is represented by the formula:

$$\text{Moment of force} = AE \times \text{weight of water.} \tag{6}$$

By using the triangles AVT and AEG and angles TAV and EAG the determination of AE is exactly the same as for AD and is found to be 35.35.

Since 100 g were taken as the load in this case, by substituting in (6) 100 for weight of water and 35.35 for AE, we get:

$$\text{Moment of force} = 35.35 \times 100 = 3535.$$

Referring to the diagram, Fig. 4, when the beam is in the horizontal position represented by BC and the pointer is at zero graduation, the moment of force is represented by the formula:

$$\text{Moment of force} = AC \times \text{weight of water.} \qquad (7)$$

Since $AC = 35.56$ and since 100 g were taken as the load in this case, by substituting in (7) 100 for weight of water and 35.56 for AC, we get:

$$\text{Moment of force} = 35.56 \times 100 = 3556.$$

Consequently, there are the following moments of force at zero graduation and at graduations numbered 1 and 2:

Moment of force at 0 = 3556;
Moment of force at 1 = 3551;
Moment of force at 2 = 3535.

From these results it can be seen that the error due to the different moments of force caused by different positions of the beam during the process of tearing is less than 1 per cent, which is very small. Very few testing instruments of greater accuracy than this are built. It is well to state here that due to the fact that at the tearing end of the beam the pull can not be exactly perpendicular to the beam in any position and that the angle of pull will vary with the position of the beam, another error exists. However, this error is very small, for the reason that the vertical plate XY (Fig. 4) which holds the test specimen of paper is situated so close to the end of the beam that there is very little clearance between the end of the beam and the vertical plate. Consequently, due to this fact and due to the fact that the angular displacement of the beam during the process of tearing is never more than 10° because the beam is so long, the angle of pull is always very close to the perpendicular. In this connection note the angles of pull, JNA, IMA, and HBA (Fig. 4), which are very close to right angles. H represents the starting point of tear, which is halfway between the end B of the beam in initial position and the lower end Y of the test sample.

3. ERROR DUE TO IMPACT OF WATER.

Let there now be taken up the second question: Are there errors caused by the force exerted by the falling stream of water? In practically all cases the paper began to tear at the balancing

position of the beam, and the water was shut off immediately by turning of the stopcock which closed the outlet from the burette, and the reading from the burette was taken. Consequently, it was decided to measure the distance from the end of the burette to the surface of the water in the glass at different applied volumes, as indicated by the graduations on the burette. This was done because the different velocities caused by the falling of the water from the burette through different distances to the surface of the water in the glass were the controlling factors in determining the error due to the impact of water. During this operation the beam was held firm in balancing position and a piece of aluminum attached to a thread was used to make the measurements. The following measurements were obtained:

Volume applied, cm^3	Distance, cm	Volume applied, cm^3	Distance, cm
25	16.5	175	11.4
75	14.6	225	10.2
125	13.0	275	8.9

Then a small piece of aluminum about the size of a 10-cent piece was attached by means of very small wires to the end of the beam in place of the glass. This was done in such a way that the water from the burette fell directly on the aluminum. The small size of the aluminum prevented any water from remaining on its surface. The forces were then measured experimentally by placing small laboratory weights on the other end of the beam to balance the force of the water from the burette on the aluminum. The forces were obtained for heads of water in the burette at different graudations (on the burette) corresponding to the above applied volumes (since the applied volumes are determined by the graduations on the burette), and the burette was lowered at the end of each experimental operation so that the distances between burette and aluminum were the same as the above at the respective applied volumes. The same operations were repeated continuously until close checks were obtained. Ten readings were taken at each head of water and an average of the 10 made. The results which were obtained are shown in Table 5. (It might be added here that the force of impact on the aluminum disk is not exactly the same as the force of impact on the water in the glass, because the energy dissipated is not the same in both cases. However, the use of the aluminum disk would seem to give results that are sufficiently accurate for a study of an instrument of this type.)

TABLE 5.—Forces at Different Heads

Readings	Forces in grams for heads at graduations of—					
	25 cm³	75 cm³	125 cm³	175 cm³	225 cm³	275 cm³
1	4.80	4.30	3.80	3.50	3.30	3.10
2	4.80	4.30	3.80	3.50	3.30	3.10
3	4.80	4.40	3.80	3.50	3.20	3.00
4	4.85	4.40	3.80	3.50	3.20	3.10
5	4.90	4.45	3.80	3.50	3.30	3.00
6	4.90	4.45	3.80	3.40	3.20	3.00
7	4.90	4.40	3.80	3.40	3.20	3.10
8	4.95	4.30	3.80	3.40	3.30	3.00
9	4.95	4.40	3.80	3.40	3.20	3.00
10	4.95	4.40	3.80	3.50	3.20	3.10
Average	4.88	4.38	3.80	3.46	3.24	3.05

From the above results a correction curve was drawn, which is presented in Fig. 5.

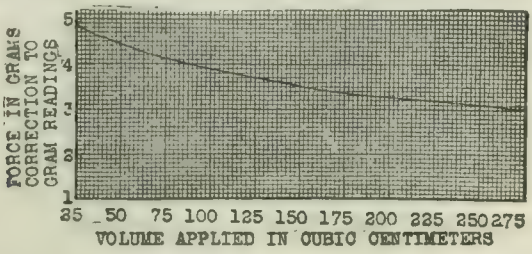

FIG. 5.—*Diagram showing positive corrections at different applied volumes as indicated by head graduations in Table 5 (nonrecording instrument, type II)*

It can easily be seen from Fig. 5 curve, by dividing the ordinates by the corresponding abscissas, that the errors caused by the force exerted by the falling stream of water are as follows:

Volume applied, cm³	Error, per cent	Volume applied, cm³	Error, per cent
25	19.52	175	1.97
75	5.84	225	1.44
125	3.04	275	1.10

Practically all the samples of bag paper that were tested on this instrument tore above the 75 g load when the test samples were cut 4 by 10 inches, which is the size recommended and specified by the inventor. This has previously been presented in Table 1. Consequently, the instrument is within the 5 per cent error on the majority of these grades of bag paper. For writing paper, such as manifold, lightweight printings, and lightweight writings, and for all lightweight short-fibered papers, the error is greater than 5 per cent, as is shown in Table 2. The term "lightweight" here indicates that the weight in pounds of the standard

size ream 25×40—500 is less than 50. For practically all writing papers that are heavier than 50 pounds, and for practically all weights of bonds and ledgers, the error would be less than 5 per cent. These errors might be decreased somewhat if the burette were lowered nearer the glass than was the case in the above experiments. However, the decrease would be very small and the results would be comparatively the same.

4. PERFORMANCE TEST

After studying this instrument from the standpoint of physics a number of samples of paper were torn on the machine in order to discover whether the instrument would check itself under standard conditions within the variation of the strength of the paper itself. Two sets of test samples, numbered 1 and 2, were prepared from the same bag papers as were used before and torn in the machine direction with very good check results, as are shown in Table 6. All test samples were cut 4 by 10 inches, and an average of 10 tests was taken for each result. From these results it can be seen that it is possible to repeat tests on the same grade of bag paper with this instrument and get check averages, and that errors due to side pull at the tear and to personal errors in stopping the flow of water are relatively very small.

TABLE 6.—Relative Effect of Use of Two Identical Sets of Test Samples on Tearing Strength of Bag Paper, Using Type II Instrument

Bag paper identification numbers	Tearing strength		Ratio of set 1 to set 2	Bag paper identification numbers	Tearing strength		Ratio of set 1 to set 2
	Set 1	Set 2			Set 1	Set 2	
	Grams	Grams	Per cent		Grams	Grams	Per cent
16 081	33.6	32.4	103.7	16 093	60.6	56.8	106.7
16 082	76.4	74.0	103.2	16 094	73.2	73.0	100.2
16 083	68.4	69.0	99.2	16 095	135.6	132.4	102.2
16 084	96.2	96.6	99.6	16 096	94.6	92.2	102.6
16 085	112.2	110.0	102.0	16 097	84.0	85.4	98.4
16 086	128.6	128.6	100.0	16 098	121.6	122.0	99.6
16 087	64.0	64.0	100.0	15 099	179.2	176.8	101.4
16 088	118.0	117.4	100.4	16 100	85.2	83.6	101.8
16 089	179.2	185.6	96.6	16 101	83.0	83.2	99.8
16 090	149.2	147.8	101.0	16 102	186.6	186.8	99.9
16 091	338.4	335.4	101.0	16 103	96.2	96.0	100.1
16 092	471.4	477.0	99.8	16 104	166.8	167.8	99.4

V. CONCLUSIONS

Conclusions to be drawn from this study of type I and type II instruments are that neither one of them has been perfected enough for general commercial use. Type I, the recording instrument as now manufactured, is neither a delicate, a sensitive, nor an accurate piece of apparatus, since the amount of friction in the pin-slot bearing and the friction between the pen point and the paper chart do not allow careful accurate calibration. Since the test results obtained by using different factors will not check, it would indicate also that there is a defect in the mechanism of the instrument. Type II, the nonrecording instrument, is a fair instrument for bag paper, since most bag papers tear above the 75 cm^3 or gram mark. However, many recommendations could be made, such as stronger clamps, a better device for cutting test samples an exact size, a better device to control the distance of tear of each sample and to keep the distance the same for all samples, and the elimination of the use of weights on the glass side of the beam, by which the force is immediately applied instead of being gradually applied as in the case of the water. The general idea of both instruments is good. The type I recording instrument gives a curve showing the maximum and minimum tearing strength of paper as well as five peaks in the curve showing the results of five initial tears. No fault can be found with a curve that represents the maximum and minimum tearing strength, and the initial tearing strength is just what is wanted for writing papers. However, there is great doubt whether the five peaks in the curve of type I instrument actually represent five initial tears. The fibers very near the edge of an angular slit in the test sample (or perhaps halfway between two angular slits between which the paper is torn) may be stronger than those fibers at the very edge. In such a case the result would be a rising curve and the peak would not represent the initial tear but the tearing strength of the fibers near the edge or halfway between two slits in the paper. During the work on this instrument it was noticed that some peaks in the curves were double-toothed or double-peaked. In this case the first peak probably more nearly represents the initial tear. This may be a very fine distinction, and yet if this recording instrument is going to be used as a laboratory instrument it must be accurate to the highest degree. The type II nonrecording instrument, on the other hand, gives merely the maximum tearing strength of paper, which is all that is necessary for a

mill test. All tests on the two instruments were made in the machine direction of the paper (the direction in which paper moves on the paper machine) for the reason that better comparative results could be obtained in this way. Most papers are much stronger in the cross direction than they are in the machine direction. Since this is true, if you attempt to tear these papers in the cross direction, the direction of the tear as a rule turns to the machine direction soon after the beginning of the tear. This change in the direction of the tear never occurs when the paper is torn in the machine direction. Since in practically all grades of paper a good tearing strength in the machine direction is just as essential as a good tearing strength in the cross direction, and since better comparative results are obtained by tearing paper in the machine direction, it would seem that all tearing tests should be made in the machine direction.

WASHINGTON, January 5, 1921.

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No. 195

ZINC CYANIDE PLATING SOLUTIONS

BY

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ZINC CYANIDE PLATING SOLUTIONS

By William Blum, F. J. Liscomb, and C. M. Carson

ABSTRACT

During the war zinc plating (or electro galvanizing) was extensively applied for the protection of steel against corrosion. For this purpose zinc sulphate or zinc cyanide solutions may be employed, but the latter produce more uniform distribution of the metal upon irregular-shaped parts. This paper describes the results of experiments upon the preparation, analysis, and operation of zinc cyanide plating solutions. It was found that in the preparation of such solutions a considerable economy could be effected by the use of zinc oxide to replace part or all of the zinc cyanide commonly used for this purpose.

I. INTRODUCTION

During the war there was a great demand for information upon zinc plating or "electro galvanizing," as it had been shown that zinc coatings furnish by far the best protection against the corrosion of steel.[1] Two types of zinc plating solutions have been used commercially, namely, the sulphate and cyanide solutions. Comparative experiments have shown that satisfactory deposists can be obtained from both sulphate and cyanide solutions, but that the latter possess the properties of "throwing" the deposit better upon irregular-shaped articles. In the studies conducted at this Bureau first consideration was therefore given to the operation of zinc cyanide solutions. The contents of this paper were published in the form of three short articles in the Monthly Review of the American Electroplaters' Society.[2] At that time it was planned to carry out further investigations upon both the cyanide and sulphate zinc solutions, and to include the results of such work in a more exhaustive publication. Owing to the fact, however, that it has been impossible to complete these proposed investigations and that it will probably not be possible to do so in the near future, this information is brought together in the form of a technologic paper in order to be readily available to those inter-

[1] See B. S. Circular No. 80, Protective Metal Coatings for the Rustproofing of Iron and Steel.
[2] November, 1918; December, 1918; and April, 1919.

ested in the subject and to assist others in carrying out further investigations on this subject.

II. GENERAL PRINCIPLES

In order to secure definite information regarding the best conditions of operation of plating solutions, it is necessary first to obtain reliable methods for preparing and analyzing solutions of any given composition. This need has been especially felt in connection with such mixtures as the zinc cyanide plating solutions in which, as will be explained shortly, a given composition can be produced by different methods, and for which the methods of analysis in ordinary use may be wholly inadequate.

When zinc cyanide $Zn(CN)_2$ is dissolved in sodium cyanide NaCN, a double compound, sodium zinc cyanide $Zn(CN)_2.2NaCN$ is formed by simple addition, which may be represented thus:

$$Zn(CN)_2 + 2NaCN = Zn(CN)_2.2NaCN \text{ or } Na_2Zn(CN)_4.$$

Similarly, when zinc hydroxide, $Zn(OH)_2$ (or zinc oxide, ZnO) is dissolved in sodium hydroxide (caustic soda, NaOH) a double compound, sodium zincate, Na_2ZnO_2 or $Zn(ONa)_2$ is formed,[3]

$$Zn(OH)_2 + 2NaOH = Zn(OH)_2.2NaOH = Zn(ONa)_2 + 2H_2O \text{ or}$$
$$ZnO + 2NaOH = Zn(ONa)_2 + H_2O.$$

It is also possible to dissolve zinc cyanide in sodium hydroxide, or to dissolve zinc oxide in sodium cyanide. In each case the same product is formed; that is, a mixture of sodium zinc cyanide and sodium zincate.

$$2Zn(CN)_2 + 4NaOH = Zn(CN)_2.2NaCN + Zn(ONa)_2 + 2H_2O$$
$$2H_2O + 2ZnO + 4NaCN = 2Zn(CN)_2 + 4NaOH = Zn(CN)_2.2NaCN + Zn(ONa)_2 + 2H_2O.$$

From these considerations, it is obvious that when, as is usually the case, it is desired to have considerable free alkali in a zinc cyanide solution, a solution identical in all respects can be prepared by the substitution of zinc oxide for all or a part of the zinc cyanide, and the use of a proportionately greater amount of sodium cyanide, and less amount of sodium hydroxide. The net saving of such

[3] Strictly speaking, the compound formed is more probably $NaHZnO_2$ or Zn(OH) (ONa). Since, however, the amount of alkali actually required to form a clear solution is at least equivalent to that in the compound $Zn(ONa)_2$, we will employ that formula for simplicity.

a substitution is represented by the economy of adding zinc in the form of a zinc oxide instead of zinc cyanide, and of adding cyanide in the form of sodium cyanide instead of zinc cyanide. The question of whether satisfactory solutions may be prepared with less total cyanide was not considered in these experiments and is not necessarily connected with such a substitution.

III. PREPARATION OF SOLUTIONS

The following approximate factors showing the relations by weight between the various constitutuents are used in the subsequent calculations:

Zinc cyanide×0.56=zinc.
Zinc cyanide×0.69=zinc oxide.
Zinc cyanide×0.68=equivalent sodium hydroxide.
Zinc cyanide×0.84=equivalent sodium cyanide.
Zinc×1.80=zinc cyanide.
Zinc×1.24=zinc oxide.
Zinc oxide×0.80=zinc.
Sodium hydroxide×1.23=equivalent sodium cyanide.
Sodium cyanide×0.83=equivalent sodium hydroxide.
Grams per liter×0.134=ounces per gallon.

When zinc oxide is dissolved in sodium cyanide an equivalent amount of sodium hydroxide, almost exactly equal to the weight of the zinc oxide used, is liberated. In order therefore to determine how much zinc cyanide in a plating solution may be substituted by zinc oxide, all that is necessary is to multiply the desired zinc cyanide content by 0.69 or for practical purposes 0.7, giving the equivalent amount of zinc oxide. If this amount is equal to or less than the desired amount of sodium hydroxide, the entire amount of zinc oxide indicated may be used. Should this amount be greater than the desired amount of sodium hydroxide, only as much zinc oxide may be used as the desired sodium hydroxide content. In the latter case sufficient zinc cyanide must also be added to furnish the desired zinc content.

The additional amount of sodium cyanide to be used is easily determined by multiplying the amount of zinc cyanide to be replaced by zinc oxide, by 0.84. The total cyanide to be used will then be equal to this plus the original content of sodium cyanide.

The amount of sodium hydroxide required is simply the originally specified amount minus the amount of zinc oxide used.

Examples:

Suppose it is desired to prepare the following two solutions by the use of zinc oxide:

TABLE 1.—Customary Formulas[4]

Ingredients	I		II	
	g/l	oz/gal	g/l	oz/gal
Zinc cyanide	75	10	75	10
Sodium cyanide	30	4	60	8
Sodium hydroxide	60	8	30	4

The amount of zinc oxide equivalent to the zinc cyanide is equal to $75 \text{ g/l} \times 0.7 = 52.5 \text{ g/l}$. The use of this amount of zinc oxide would liberate an equal amount of sodium hydroxide. In solution I, where 60 g/l of sodium hydroxide is desired, the full amount of sodium hydroxide will be required. In solution II where it is desired to have 30 g/l of sodium hydroxide, only that amount—that is, 30 g/l of zinc oxide—may be used instead of 52.5 g/l. Consequently only $\frac{30}{52.5} \times 75 = 43$ g/l of the zinc cyanide can be substituted, leaving 32 g/l of zinc cyanide to be added. The amount of sodium cyanide to be used in addition to that originally specified is 0.84 times the amount of zinc cyanide replaced; that is, $0.84 \times 75 = 63$ g/l in solution I and $0.84 \times 43 = 36$ g/l in solution II. The revised formulas will then be as follows:

TABLE 2.—Formulas Using Zinc Oxide

Ingredients	I		II	
	g/l	oz/gal	g/l	oz/gal
Zinc cyanide	0.	0.	32	4.3
Zinc oxide	52.5	7.	30	4.0
Sodium cyanide	93.	12.4	96	12.8
Sodium hydroxide	7.5	1.0	0	0

In the preparation of such solutions it is necessary to first dissolve the sodium cyanide and hydroxide in one-half or less of the final volume of water and then to slowly pour the zinc oxide, previously mixed with water to form a thin paste, into the first solution while stirring. When the solution is complete, addi-

[4] In these formulas and calculations no attention has been paid to the presence of minor constituents, such as aluminum sulphate, tartrates, etc., which are sometimes used in these solutions. For simplicity the calculations are made only in terms of the metric units, although obviously exactly the same calculations can be made with ordinary units.

tional water may be added to produce the desired volume or specific gravity.

The exact economy to be effected by such substitutions will depend upon the market prices of the chemicals involved, and can be readily computed for such examples as are given. In this connection it should be borne in mind that zinc oxide contains 80 per cent of metallic zinc as against 56 per cent in zinc cyanide; and that sodium cyanide contains 53 per cent of cyanogen (CN) as against 45 per cent in zinc cyanide. Solutions prepared by this method have been found to produce results equal to those obtained in solutions of similar composition prepared from zinc cyanide.

IV. ANALYSIS OF SOLUTIONS

1. PRINCIPLES

While the methods for the determination of zinc in plating solutions are comparatively simple and reliable, the methods frequently employed for the determination of free cyanide and free alkali yield such erratic results that they are of little or no value. The principal cause of uncertainty in such methods is that they attempt to determine two quantities which vary with dilution and which indeed can scarcely be said to exist. By the methods to be described in this paper it is possible to determine the three principal constituents, viz, total zinc, total cyanide, and total alkali. From these values may be calculated, if desired, the excess of one or another constituent, although for control purposes such calculations may not be necessary.

As explained above, the principal compounds existing in zinc cyanide plating solutions are sodium zinc cyanide $Zn(CN)_2$. $2NaCN$ and sodium zincate $Zn(ONa)_2$, together with any excess of sodium cyanide or sodium hydroxide that may be present. If such compounds were stable—that is, difficulty decomposable—the analysis of such solutions would be comparatively simple. As a matter of fact, however, they are readily dissociated or decomposed by the action of water itself so that even dilution will change their relative amounts.

This decomposition may take place from either one or both of two causes, viz, dissociation and hydrolysis, both of which are reversible; that is, the reactions may take place in either direction. The extent to which such changes take place is determined by the various conditions such as concentration (or dilution), temperature, etc.

"Dissociation" is simply the breaking up of a compound into elements or simpler compounds. For example, if it were possible to prepare in a dry state sodium zinc cyanide, having the formula $Zn(CN)_2.2NaCN$, this compound when dissolved in water would decompose in part into zinc cyanide and sodium cyanide, the decomposition taking place to a greater extent the greater the proportion of water used. If more sodium cyanide were then added to the solution, the reaction would be reversed and more of the double compound would be formed. Conversely, if some of the sodium cyanide first liberated is removed from the solution, the decomposition continues and more sodium cyanide is set free. The above facts are all expressed in the reversible chemical equation.

$$Zn(CN)_2.2NaCN \rightleftharpoons Zn(CN)_2 + 2NaCN,$$

in which the double arrow shows that, depending upon the conditions, the reaction may take place in either direction.

"Hydrolysis" is the decomposition of a compound brought about by the presence of water, which actually enters into the chemical reaction which takes place. As is well known, when an acid and a base react with each other a salt and water are formed. Salts formed from a weak acid and a strong base, or vice versa, when dissolved in water are decomposed to some extent by the water into the original acid and the base. These are liberated in equivalent quantities, but if the base is stronger than the acid, the solution is alkaline toward the customary indicators such as litmus, phenol-phthalein, etc.; while if the acid is stronger, the solution is acid to indicators. Thus when sodium cyanide is dissolved in water the solution is alkaline, because (chemically) hydrocyanic acid, HCN, is a weak acid, while sodium hydroxide is a strong base. The reaction may be expressed thus:

$$NaCN + H_2O \rightleftharpoons NaOH + HCN.$$

Similarly sodium zincate when dissolved in water is alkaline, because zinc hydroxide behaves as a weak acid in such salts as sodium zincate.

If, therefore, an attempt is made to determine the excess of sodium cyanide in a zinc solution—for example, by titration with silver nitrate—more cyanide is liberated as the free cyanide is used up by the silver nitrate. Similar results are obtained if an attempt is made to titrate the free alkali directly with acid.

Reliable determinations can be made, however, of the total cyanide and total alkali, which for convenience may be expressed as sodium cyanide and sodium hydroxide, respectively, though they may not have been added in that form. Omitting any detailed explanation of the principles involved, the methods for the above determinations may be briefly described as follows. These determinations may readily be carried out by anyone with moderate chemical training and laboratory equipment. They are based in part upon methods described in Clennel's Chemistry of Cyanide Solutions, and Williams' Chemistry and Manufacture of Cyanides.

2. METHODS OF ANALYSIS

(*a*) *Preparation of the Sample.*—The solution should be filtered if turbid, and a measured volume (for example, 25 cc) should be diluted to a definite volume (for example, 500 cc) in a graduated flask. This diluted solution is used for the determination of total cyanide and total alkali. The zinc may be most conveniently determined on a separate sample of the original solution.

(*b*) *Total Zinc.*—To 2.5 cc of the original solution (or 50 cc of the diluted solution), add 5 cc each of concentrated nitric acid and concentrated sulphuric acid and evaporate the solution till dense white fumes appear. This evaporation must be conducted in a well-ventilated hood. After cooling the residue add 3 cc of concentrated hydrochloric acid and 20 cc of water. The solution should then show no trace of blue color. If it does, add a few cc of nitric acid and again evaporate the solution. Precipitate any iron and aluminum that may be present by the addition of ammonium hydroxide until the solution is just alkaline to litmus, and filter out and wash the precipitate, which may be used for the determination of iron and aluminum in the regular way.

Acidify the filtrate and determine the zinc in it by titration with potassium ferrocyanide, using uranium acetate as an outside indicator. Details of this titration may be found in any of the standard books on quantitative analysis. From the strength of the ferrocyanide solution (as determined by standardization against a definite amount of pure zinc) and the amount required for the titration, the zinc content of the solution may be readily calculated.

(*c*) *Total Cyanide.*—Dilute 25 cc of the previously diluted solution to 200 cc with distilled water, and add approximately 5

cc of 10 per cent sodium hydroxide solution and 1 cc of 10 per cent potassium iodide solution. Titrate the solution with tenth normal silver nitrate solution until a permanent faint yellow turbidity is produced. The solution should be stirred vigorously during the titration. Add a little more sodium hydroxide, and if the turbidity disappears, continue the titration with silver nitrate until a permanent turbidity is produced. The silver nitrate solution may be prepared by dissolving exactly 17 g of silver nitrate in distilled water and diluting to 1 liter. 1 cc of this solution = 0.0098 g sodium cyanide, NaCN.

This titration gives the total cyanide including that originally added both in the form of zinc cyanide and sodium cyanide. It is most convenient, however, to express the result as sodium cyanide, as will be illustrated in the subsequent examples. The results by this titration are not affected by the presence of aluminum salts or tartrates, but are too high in the presence of ammonium hydroxide. Addition of ammonium hydroxide, as frequently recommended for free cyanide titrations, should therefore be avoided.

(*d*) *Total Alkali.*—The method which gives best results, which, however, are only approximate, is as follows:

Dilute 25 cc of the diluted solution to 200 cc, and add slightly more than enough potassium ferrocyanide solution to precipitate the zinc, that is, slightly more than one-half of that required in (*b*), and slightly more silver nitrate than the amount required in (*c*). Then add a few drops of phenolphthalein solution, and titrate with standard acid (fifth normal hydrochloric acid) until the pink color disappears for at least a few minutes.

By this method the total alkaline hydroxide plus one-half the carbonate is obtained. The amount of carbonate present may be separately determined by precipitation with barium hydroxide and an appropriate correction made. Such a correction is necessary only if zinc carbonate or sodium carbonate has been added to the solution or if the solution has been in use for a long period. In most cases for control work the total alkali determined may be calculated as sodium hydroxide.

3. CALCULATION OF THE RESULTS

An analysis of a plating solution may be conducted for either one or both of two purposes, first, in order to duplicate the solution or, second, to determine what changes have occurred during

operation, and what additions (or removals) may be necessary in order to restore the solution to the original or to any other desired composition. In the former case it is necessary to calculate the amounts of the actual ingredients to be used in preparing the solution, while in the latter the direct results of the analysis may furnish the desired information.

It should be noted that in many cases it is impossible to state what amounts of the various possible compounds are actually present in solution. Thus, for instance, in a zinc cyanide solution containing a total amount of sodium cyanide and sodium hydroxide greater than is required to combine with the zinc, it is impossible to state whether the zinc is combined with the sodium cyanide, leaving the hydroxide in excess, or vice versa. In short, for such solutions, any results that may be obtained or calculated for "free cyanide" or "free alkali" are entirely arbitrary and may be very misleading. At the same time it may be very useful to determine for the sake of comparison what is the total excess of sodium cyanide and hydroxide over that required to combine with the zinc.

In order to determine the ingredients—that is, the amounts of materials required to prepare such a solution—it is necessary first to decide whether the zinc is to be added entirely as zinc cyanide, or, so far as possible, as zinc oxide. In the former case, the steps in the calculation are as follows:[5]

(*a*) Multiply the zinc content by 1.8 to obtain the zinc cyanide required.

(*b*) Multiply the zinc cyanide content thus found by 0.84 to obtain the equivalent amount of sodium cyanide.

(*c*) Deduct the amount of sodium cyanide found in (*b*) from the total cyanide to obtain the amount of sodium cyanide to be used.

(*d*) The amount of sodium hydroxide to be used is equal to that found in the analysis.

In case zinc oxide is to be substituted for zinc cyanide, the calculations are as follows:

(*e*) Multiply the zinc content by 1.24 to obtain the equivalent of zinc oxide. If this result is equal to or less than the amount of sodium hydroxide as indicated by the titration, the entire amount of zinc oxide may be used. Otherwise the amount of zinc oxide to be used is equal to the amount of sodium hydroxide desired.

[5] Since the results of analysis are always obtained in metric units, for instance in grams per liter, g/l, the calculations may be carried out most conveniently in such units, and the final results transposed to ounces per gallon, if desired, by multiplying them by 0.134.

(*f*) If all the zinc may be added as zinc oxide, no zinc cyanide is required. If, as indicated by the calculation in (*e*), only part of the zinc is to be added as zinc oxide, multiply the amount of zinc oxide to be used by 0.80 to obtain its equivalent of zinc. Deduct this from the total zinc content, and multiply the remaining zinc by 1.8 to obtain the zinc cyanide required.

(*g*) If any zinc cyanide is to be used, multiply its amount by 0.84 to obtain the equivalent sodium cyanide, and deduct this from the total cyanide to obtain the amount of sodium cyanide to be added.

(*h*) From the total sodium hydroxide found by analysis deduct the amount of zinc oxide used, leaving the amount of sodium hydroxide to be added.

As previously stated, it is impossible to calculate the amounts of sodium zinc cyanide and sodium zincate present in such solutions or the actual excess of sodium cyanide or sodium hydroxide. It is a simple matter, however, to determine and express arbitrarily in terms of sodium cyanide or sodium hydroxide the excess of these two substances over that required to combine with the zinc. For convenience, we may consider the cyanide as combined with the zinc to form sodium zinc cyanide, and any balance of the zinc as combined with the sodium hydroxide, leaving the latter in excess. The calculation would then be as follows:

(*i*) Multiply the total cyanide (expressed as sodium cyanide) by 0.33 to obtain the equivalent amount of zinc. If this amount of zinc is less than the total zinc, deduct it from the total zinc to determine the amount of zinc combined with sodium hydroxide. If the amount of zinc present is less than that above calculated, multiply the total zinc content by 3 to obtain the sodium cyanide required to combine with it. The balance of the sodium cyanide may then be considered as "excess cyanide."

(*j*) Multiply the amount of zinc uncombined with sodium cyanide, as found in (*a*), by 2.45 to obtain the amount of sodium hydroxide required to combine with that amount of zinc. The amount of sodium hydroxide present above that amount may be considered as "excess sodium hydroxide." Obviously by this arbitrary method of calculation, if there is any "excess cyanide" present, all the sodium hydroxide is "excess sodium hydroxide."

All the principles and methods above described may be illustrated by means of two zinc cyanide solutions, which throughout will be designated as No. I and No. II.

Total Zinc.—2.5 cc of the solution (or 1/400 liter) were treated and titrated as described with a potassium ferrocyanide solution (of which 1 cc was equal to 0.005 g of zinc).

No. I required 20 cc and No. II required 17 cc of the ferrocyanide.

No. I. $20 \times 0.005 \times 400 = 40$ g/l zinc.
No. II. $17 \times 0.005 \times 400 = 34$ g/l zinc.

Total Cyanide.—25 cc of the original solution was diluted to 500 cc, and 25 cc of the diluted solution (that is, 1.25 cc or 1/800 liter of the original) was used in the titration with tenth normal silver nitrate (1 cc of which is equal to 0.098 g NaCN).

No. I required 17 cc and No. II required 10 cc of silver nitrate.

No. I. $17 \times 0.0098 \times 800 = 133$ g/l sodium cyanide.
No. II. $10 \times 0.0098 \times 800 = 78$ g/l sodium cyanide.

Total Hydroxide.—For 25 cc of the diluted solution (that is, 1/800 liter) were added to No. I about 12 cc of the ferrocyanide solution and about 18 cc of silver nitrate, and to No. II about 10 cc of the ferrocyanide and about 11 cc of silver nitrate; that is, in each case a few cc more of each solution than was required for the same amount of sample in the first two titrations. The solutions were then titrated, after the addition of phenol-phthalein, with fifth normal hydrochloric acid (1 cc of which is equal to 0.008 g sodium hydroxide).

No. I required 6 cc and No. II required 11 cc of hydrochloric acid.

No. I. $6 \times 0.008 \times 800 = 38$ g/l sodium hydroxide.
No. II. $11 \times 0.008 \times 800 = 70$ g/l sodium hydroxide.

The two solutions were therefore found by analysis to contain:

Ingredients	No. I: g/l	No. II: g/l
Total zinc	40	34
Total cyanide (expressed as sodium cyanide)	133	78
Total hydroxide (expressed as sodium hydroxide)	38	70

To prepare these solutions from zinc cyanide will require:

No. I. $40 \times 1.8 = 72$ g/l zinc cyanide.
No. II. $34 \times 1.8 = 61$ g/l zinc cyanide.

These are equivalent to:

No. I. $72 \times 0.84 = 61$ g/l sodium cyanide.
No. II. $61 \times 0.84 = 51$ g/l sodium cyanide.

Therefore the solutions require:

No. I. $133 - 61 = 72$ g/l sodium cyanide.
No. II. $78 - 51 = 27$ g/l sodium cyanide.

And the full amounts of sodium hydroxide, viz, 38 g/l and 70 g/l, respectively.

To prepare them by the use of zinc oxide would require:

No. I. $40 \times 1.24 = 50$ g/l zinc oxide.
No. II. $34 \times 1.24 = 42$ g/l zinc oxide.

In No. I, however, only 38 g/l of sodium hydroxide is present; therefore only 38 g/l of zinc oxide may be used. This is equivalent to $38 \times 0.8 = 30$ g/l zinc, leaving 10 g/l of zinc to be added as zinc cyanide; that is, $10 \times 1.8 = 18$ g/l of zinc cyanide, which is equivalent to $18 \times 0.84 = 15$ g/l of sodium cyanide; therefore $133 - 15 = 118$ g/l of sodium cyanide is required; while in No. II, where all the zinc is to be added as zinc oxide, the full amount of sodium cyanide, 78 g/l, is required. In No. I no sodium hydroxide will be required (since it is all supplied by the solution of the zinc oxide); while in No. II, $70 - 42 = 28$ g/l of sodium hydroxide is required. The formulas for the preparation of the solutions will therefore be as follows:

WITH ZINC CYANIDE

Ingredients	No. I		No. II	
	g/l	oz/gal	g/l	oz/gal
Zinc cyanide	72	9.6	61	8.2
Sodium cyanide	72	9.6	27	3.6
Sodium hydroxide	38	5.1	70	9.4

WITH ZINC OXIDE

Zinc oxide	38	5.1	42	5.6
Zinc cyanide	18	2.4	0	.0
Sodium cyanide	118	15.8	78	10.4
Sodium hydroxide	0	.0	28	3.7

The calculation of assumed excess cyanide may be made as follows:

No. I. The total cyanide is equivalent to 133 g/l × 0.33 = 44 g/l zinc. Since this is greater than the zinc content (40 g/l) there must be an excess of cyanide. 40 g/l × 3 = 120 g/l sodium cyanide, therefore there is 133 − 120 = 13 g/l excess sodium cyanide; and the entire amount, that is, 38 g/l, excess sodium hydroxide.

No. II. The total cyanide is equivalent to 78 g/l × 0.33 = 26 g/l of zinc. Therefore 34 − 26 = 8 g/l of zinc is combined with sodium hydroxide. 8 × 2.45 = 20 g/l of sodium hydroxide is required to combine with 8 g/l of zinc. The solution therefore contains 70 − 20 = 50 g/l excess sodium hydroxide.

The solutions may therefore be said to contain:

Ingredients	No. I		No. II	
	g/l	oz/gal	g/l	oz/gal
Excess sodium cyanide	13	1.7	0	0.0
Excess sodium hydroxide	38	5.1	50	6.7

V. OPERATION OF SOLUTIONS

Since, as shown previously, it is possible to replace zinc cyanide in the customary plating formulas by zinc oxide in amounts up to the desired content of sodium hydroxide, and since in preliminary experiments it was found that a rather high content of sodium hydroxide was desirable, all the solutions employed in these experiments were prepared by the solution of zinc oxide in sodium cyanide, with in some cases the actual addition of sodium hydroxide. It should be clearly understood, however, that any of these solutions may be prepared by the use of zinc cyanide instead of zinc oxide, with appropriate changes in the amounts of the other constituents. The composition recommended will therefore be expressed in terms of the two methods of preparation.

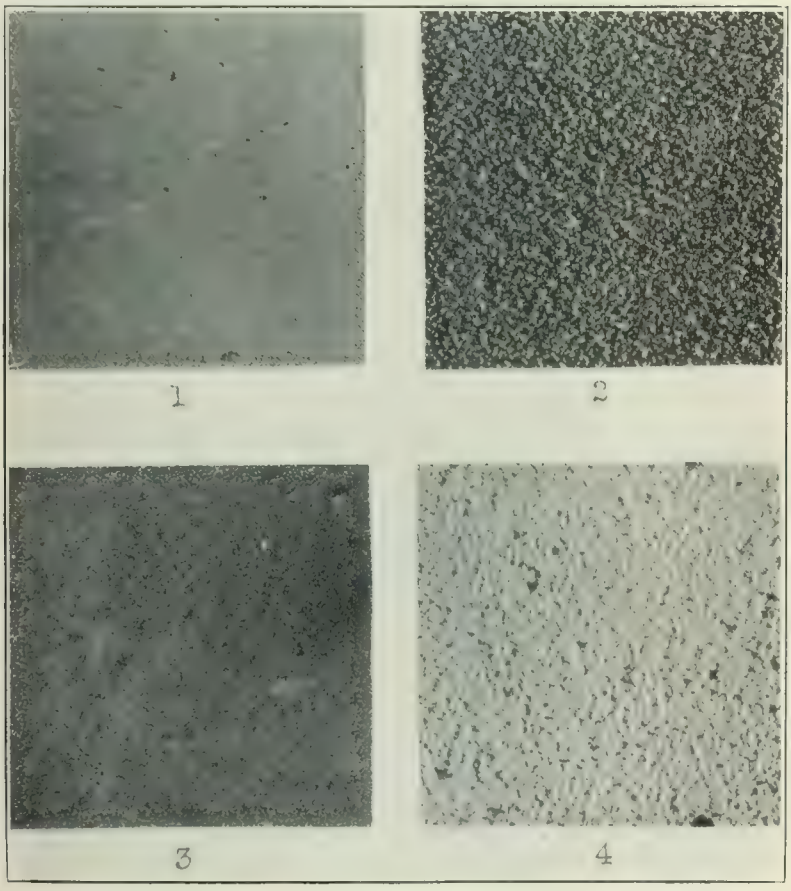
Preliminary experiments upon the solubility of zinc oxide in sodium cyanide showed that although according to the simple equations 81 parts of zinc oxide should dissolve in 98 parts of sodium cyanide—that is in 1.21 times its own weight—considerably more than this proportion of sodium cyanide is actually required to produce a clear solution. This observation is in accordance with the well-known fact that sodium zincate solutions require a considerable excess of alkali, and sodium zinc cyanide

solutions an appreciable excess of cyanide to overcome the effects of hydrolysis and dissociation, and to maintain clear solutions. For practical purposes it was found that to produce complete solution the sodium cyanide must be about 1.67 times the weight of the zinc oxide. Additional cyanide and hydroxide were added in some cases.

The effects of composition, concentration, current density, and temperature were determined in a large number of experiments, usually conducted in baths containing about 1200 cc and operated in series with a copper voltameter. Observations were also made upon the conductivity of the solutions, the polarization occurring under the conditions employed, the anode and cathode efficiences, and the appearance and microstructure of the deposits. In the selection of favorable operating conditions, emphasis has been placed upon the structure of the deposits since, other things being equal, the best conditions are those that lead to a fine-grained uniform deposit which will, for a given weight of zinc, exert the most protective action upon the steel. To illustrate the characteristic features of the structure, a magnification of at least 20 diameters is necessary. Even with a pocket lens, however, it is easily possible to distinguish between deposits with fine and coarse crystals, and between those having a dense structure and those having a loose or porous structure. A dark or powdery appearance to the naked eye is evidence of a poor deposit. On the other hand, the "whitest" deposits are not necessarily the most dense. Usually the fine dense deposits have a light silvery luster.

The following conclusions may be drawn from the experiments thus far conducted at this Bureau:

1. *Composition and Concentration.*—As previously stated, it was found necessary to have 1.67 times as much sodium cyanide as zinc oxide; for example, for 45 g/l (6 oz./gal.) of zinc oxide, 75 g/l (10 oz./gal.) of sodium cyanide are required. Any excess of cyanide above this amount—for example, an excess of 15 g/l (2 oz./gal.)—in every case produced coarse deposits (Fig. 2). Fine-grained deposits (Fig. 1) can be secured from solutions containing only zinc oxide and sodium cyanide in the above proportions, but to do so requires a relatively high voltage, especially with current densities higher than 1 amp/dm^2 (9.3 amp/sq. ft.). This was found to be due not so much to a low conductivity of such solutions as to a high polarization, arising from accumulation of insoluble products upon the anodes. This condition was imme-



FIGS. 1 to 4

diately remedied by the addition of small amounts of sodium hydroxide, up to 15 g/l (2 oz./gal.). From such solutions good deposits (Fig. 3) were obtained over a wide range of conditions.

In general, with the above proportion of the constituents there is not much effect due to concentration. In dilute solutions at low current densities, or for very thin deposits (as, for example, in a cyanide "strike" solution) the deposits appear to be somewhat finer grained than in the more concentrated solutions. On the other hand, in more concentrated solutions the resistance and polarization are lower, and higher current densities can be employed (for example, up to 3 amp/dm^2 or 28 amp/sq. ft.). The use of fairly concentrated solutions, containing at least 45 g/l (6 oz./gal.) of zinc oxide, or 36 g/l (4.8 oz./gal.) of metallic zinc, is therefore recommended. There is a decided disadvantage in the presence of carbonates, which should not therefore be introduced into the solution, either by the addition of zinc carbonate or sodium carbonate, or by the use of sodium cyanide or hydroxide containing much carbonate. In general, carbonates tend to produce coarse or striated deposits (Fig. 4). Since some carbonate always forms in such solutions, owing to absorption of carbon dioxide from the air, and to decomposition of the cyanide, it is desirable to remove it periodically, for example, every three or six months at the time of the regular "clean-up." This can be readily accomplished by the addition of barium cyanide solution, or equally well of a solution containing sodium cyanide and barium hydroxide. The precipitate of barium carbonate which separates should be allowed to settle, after which the clear solution may be siphoned out in the usual way. Carbonates can also be eliminated to a great extent by simply cooling the solution to 0° C or 32° F, or to as low a temperature as possible, when sodium carbonate crystallizes out and can be readily removed. While it is not possible to state just what is the maximum permissible content of sodium carbonate, or just how fast it will accumulate in the solution, it is desirable, if possible, to maintain it below 15 g/l (2 oz./gal.).

2. *Current Density.*—In general, an increase in the current density up to the point at which the current efficiency falls off, and marked "gassing" occurs, produces a somewhat finer structure. The effects of current densities from 1 to 3 amp/dm^2 (9 to 28 amp/sq. ft.) are not, however, sufficiently marked to warrant the use of the higher current density except from the obvious advantage of reducing the time required. For average purposes

a current density of about 2 amp/dm² (19 amp/sq. ft.) will be satisfactory; on plain objects higher current densities may be employed and for complicated shapes a lower current density must be used to prevent excessive deposition on edges, etc.

3. *Temperature.*—In general, it is found that warm solutions produce somewhat coarser deposits than cold solutions. On the other hand, they permit good deposits to be secured at somewhat higher current densities, partly because any gas liberated on the cathode is less likely to adhere at the higher temperatures. The better conductivity of the warm solution permits the use of a lower voltage than in cold solutions to secure any desired current density. For most purposes where there is no necessity for using current densities above 2 amp/dm² (19 amp/sq. ft.) there is, however, little if any advantage in heating the solutions. In any case the temperature should not be allowed to rise above 50° C (122° F); otherwise excessive decomposition of cyanide occurs.

4. *Conditions Recommended.*—In suggesting the following conditions of operation it is not implied that these are necessarily the best for practical operation for any given class of work. It is believed, however, that these conditions will serve as a guide in the preparation, operation, and control of such solution. To avoid confusion a definite formula is given, although as previously stated the composition may vary somewhat without producing noticeable effects. For purposes of comparison the concentrations are expressed in terms of the normality, but for convenience of application the values have been rounded off to be equivalent to a simple number of ounces per gallon, making allowances for amounts of impurities likely to be present in the commercial chemicals.

1. Composition.

Ingredients	N	g/l	oz/gal
Zinc oxide	1.0	45	6
Sodium cyanide	1.5	75	10
Sodium hydroxide	.3	15	2

The same solution can be prepared from:

Ingredients	N	g/l	oz/gal
Zinc cyanide	1.0	65	8.7
Sodium cyanide	.3	20	2.7
Sodium hydroxide	1.5	60	8

2. Temperature: 20 to 40° C (68 to 104° F.).

3. Current density: Average about 2 amp/dm² (19 amp/sq. ft.).

In using zinc oxide it is desirable to obtain material nearly free from lead, any large amount of which is objectionable in the solutions. In case lead is present it will produce a gray sponge upon the anodes on standing overnight. In such a case a large number of zinc anodes should be suspended in the solution (and cleaned at intervals) until no more lead appears upon them. By this means all the lead can be removed.

In conclusion, the author desires to acknowledge the assistance of A. D. Bell, L. M. Ritchie, L. B. Ham, and S. C. Langdon in connection with these experiments.

WASHINGTON, June 9, 1921.

DEPARTMENT OF COMMERCE

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OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 196

HIGH FIRE PORCELAIN GLAZES

BY

H. H. SORTWELL, Assistant Ceramic Engineer

Bureau of Standards

SEPTEMBER 2, 1921

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1921

HIGH FIRE PORCELAIN GLAZES

By H. H. Sortwell

A great deal of material has been published on porcelains of all types, but systematic information on porcelain glazes fired at the higher temperatures is more limited.

Stull [1] thoroughly covered the field for glazes maturing at cone 11. Stull and Howat [2] investigated the field at cone 9 and reported deformation temperature studies for the glazes made.

Since no detailed information was to be had on glazes of this type for high fire porcelain, chemical porcelain, spark plugs, pyrometer tubes, etc., maturing from cones 12 to 16, the following investigation was undertaken.

Compositions Studied.—As originally laid out the field consisted of 35 members with 3, 4, 5, 7, 9, 11, and 13 equivalents of silica, and alumina corresponding to alumina-silica ratios of 1:10, 1:8, 1:6, 1:5, and 1:4. The RO members were constant at 0.7 C_aO and 0.3 K_2O. This covered the field with fair uniformity when plotted both by the empirical formula method and by batch weights. Later, to round out the deformation temperature study, 15 additional glazes were made with silica 4, 5, 7, 9, 11, and 13 equivalents with the alumina-silica ratio of 1:12; silica 5, 7, 9, 11, and 13 equivalents with the alumina-silica ratio of 1:15; and silica 7, 9, 11, and 13 equivalents with the alumina-silica ratio of 1:20.

Procedure.—The glazes were weighed out separately in 1 kg batches and ground three hours in small ball mills. The materials used were analyzed and the results given in Table 1.

A small quantity of each glaze was evaporated to dryness and molded into test cones for the deformation temperature determinations, which were made in a small gas furnace. The temperature was raised to 1100° C in one hour and from 1100° C on at the rate of 50° per hour. The temperature measurements were made with a platinum-rhodium thermocouple and a Leeds-Northrup potentiometer indicator. Five cones were tested at a time, the thermo-

[1] Trans. Amer. Cer. Soc., 14, pp. 62–70.

[2] Trans. Amer. Cer. Soc., 16, pp. 454–460.

couple junction being within three-eighths inch of each cone. Uniform temperatures were maintained within the furnace as shown by Orton cones. Two or more determinations were made on each glaze, checks within 5° C being obtained.

TABLE 1.—Results of Analyses of the Material Used

Components	Flint	North Carolina kaolin	Maine feldspar	Whiting
SiO_2	98.43	41.96	72.85	0.91
Fe_2O_3	.26	.52	.48	.30
Al_2O_5	.80	35.95	15.45	.34
CaO	.....	.24	.31	55.02
K_2O	.....	.....	7.89	.....
Na_2O	.....	.....	2.31	.....
MgO	.....	.....	.....	.44
Loss on ignition	.14	20.40	.25	43.44
Total	99.63	99.07	99.54	100.45

The glazes were applied by dipping on porcelain cups 2½ inches in diameter, 2½ inches deep, and one-eighth inch thick, which had previously been biscuited to cone 08 in a slow oxidizing fire. The composition of the body used was as follows:

North Carolina kaolin	31
Florida kaolin	12
Tennessee ball clay No. 5	5
Flint	34
Feldspar	18

Four glaze burns were made in a gas-fired kiln, to cone 10 in 20 hours, to cone 12 in 20 hours, to cone 14 in 22 hours, and to cone 16 in 24 hours. Reducing conditions were maintained up to cone 8, but from that point on reduction was diminished until the fire was neutral or slightly oxidizing at the finish.

Deformation Temperatures.—The silica and alumina equivalents of all the glazes and their deformation temperatures are given in Table 2. The deformation eutectic axis in the part of the field previously covered practically agrees with the work of Stull and Howat. The softening temperatures found in the present work are lower, as would be expected, due to the slower rate of heating. Since the field was not covered as closely in this work, the slight variations are not shown.

TABLE 2

No.	Al_2O_3	SiO_2	Al_2O_3: SiO_2	Deformation temperature, °C.	No.	Al_2O_3	SiO_2	Al_2O_3: SiO_2	Deformation temperature, °C.
1	0.300	3.0	1:10	1173	26	1.100	11.0	1:10	1275
2	.375	3.0	1:8	1146	27	1.375	11.0	1:8	1278
3	.500	3.0	1:6	1109	28	1.833	11.0	1:6	1300
4	.600	3.0	1:5	1109	29	2.200	11.0	1:5	1350
5	.750	3.0	1:4	1134	30	2.750	11.0	1:4	1418
6	.400	4.0	1:10	1137	31	1.300	13.0	1:10	1315
7	.500	4.0	1:8	1115	32	1.625	13.0	1:8	1317
8	.666	4.0	1:5	1114	33	2.166	13.0	1:6	1336
9	.800	4.0	1:6	1124	34	2.600	13.0	1:5	1386
10	1.000	4.0	1:4	1210	35	3.250	13.0	1:4	1461
11	.500	5.0	1:10	1124	A_1	.333	4.0	1:12	1175
12	.625	5.0	1:8	1124	A_2	.416	5.0	1:12	1151
13	.833	5.0	1:6	1157	A_3	.583	7.0	1:12	1148
14	1.000	5.0	1:5	1173	A_4	.750	9.0	1:12	1183
15	1.250	5.0	1:4	1223	A_5	.916	11.0	1:12	1252
16	.700	7.0	1:10	1147	A_6	1.083	13.0	1:12	1317
17	.875	7.0	1:8	1159	B_1	.333	5.0	1:15	1169
18	1.166	7.0	1:6	1208	B_2	.466	7.0	1:15	1149
19	1.400	7.0	1:5	1218	B_3	.600	9.0	1:15	1180
20	1.750	7.0	1:4	1292	B_4	.733	11.0	1:15	1230
21	.900	9.0	1:10	1209	B_5	.867	13.0	1:15	1376
22	1.125	9.0	1:8	1224	C_1	.350	7.0	1:20	1207
23	1.500	9.0	1:6	1230	C_2	.450	9.0	1:20	1237
24	1.800	9.0	1:5	1248	C_3	.550	11.0	1:20	1398
25	2.250	9.0	1:4	1370	C_4	.650	13.0	1:20	1410

The results are plotted for the molecular variations of alumina and silica (Fig. 1) and for the batch weights (Fig. 2), for which

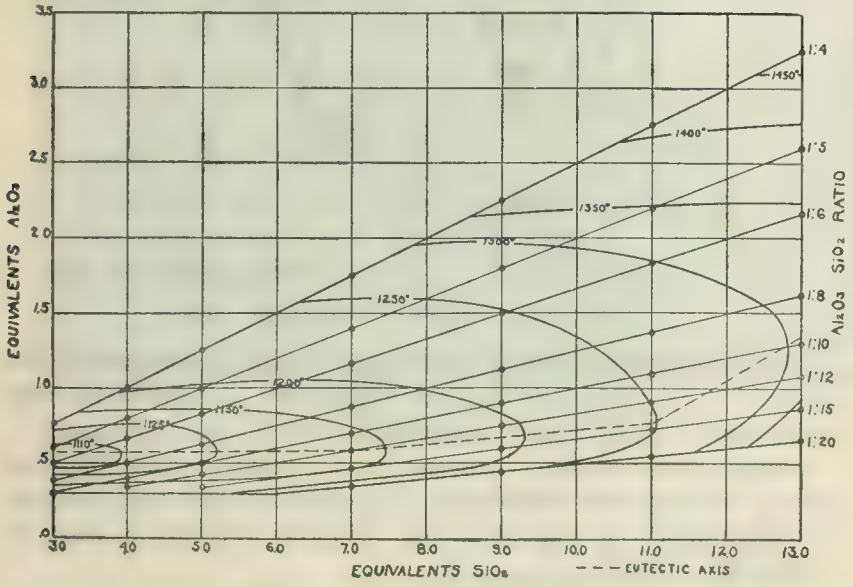


FIG. 1.—*Deformation temperatures*

the glazes were calculated into percentages of clay, flint, and fluxes (feldspar and whiting in the proportion of 167 feldspar to 70 whiting).

As shown in Fig. 1, increase in silica to 11 equivalents maintaining the alumina constant has little effect in comparison with alumina in increasing the deformation temperature as the isotherms run in the general direction of the alumina ordinate up to 11 equivalents. Beyond this figure increase in silica has a more pronounced effect on the deformation temperature.

Starting with 0.3 Al_2O_3 and any silica content, increasing the Al_2O_3 lowers the deformation temperature until the eutectic axis is crossed. Further increase in Al_2O_3 raises the deformation temperature.

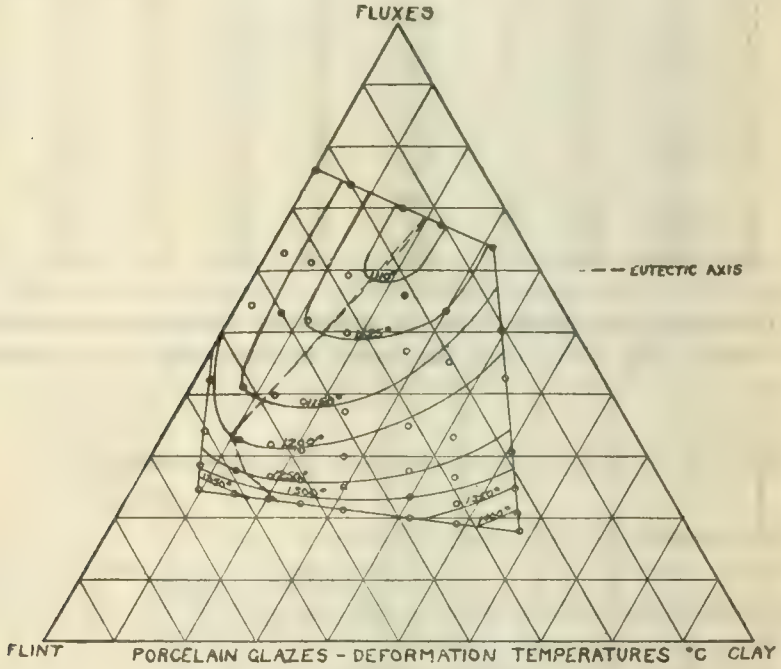


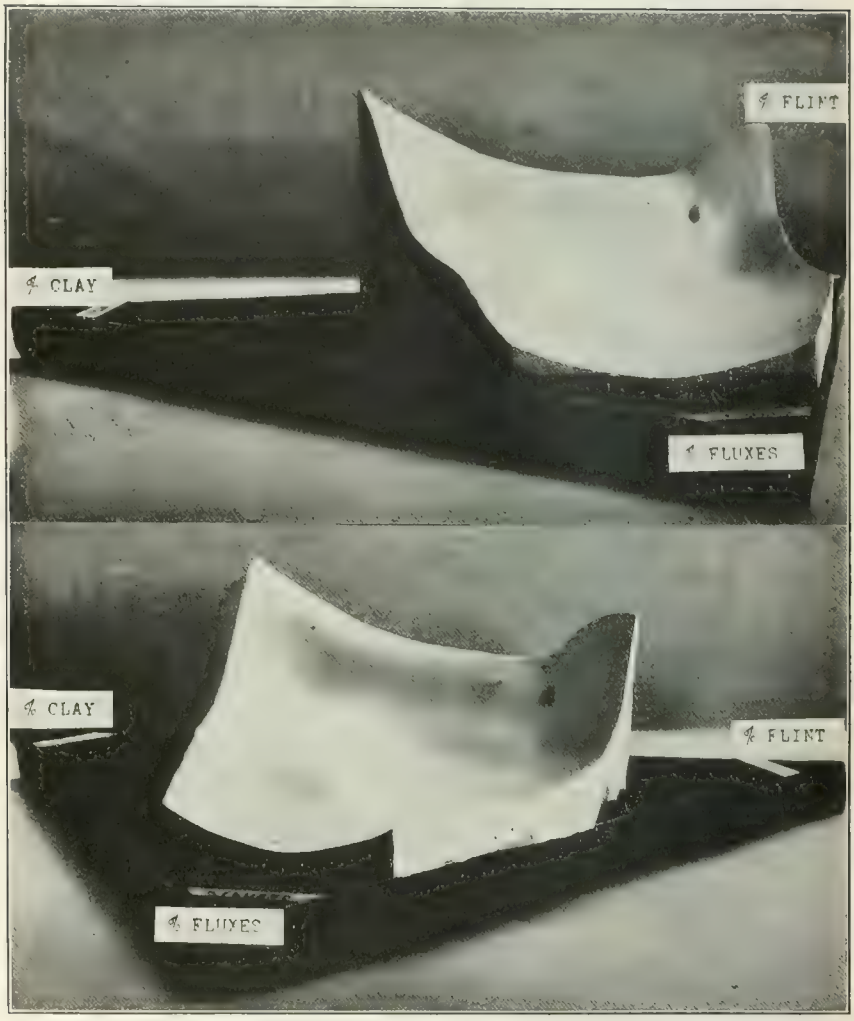
FIG. 2.—*Deformation temperatures*

The deformation eutectic axis lies at a slight inclination to the ordinate. As the silica content approaches 7 equivalents this inclination rises, and at 11 equivalents it increases much more.

Examination of Fig. 2 shows that increasing flint does not greatly increase the deformation temperature until the composition approaches 60 per cent flint. Further increase in flint increases the temperature rapidly.

Starting with zero clay, additions of clay first lower the deformation temperature, after which further increments of clay increase the temperature gradually until 45 per cent clay is ap-

FIG. 3.—*Solid diagram of deformation temperatures*



FIGS. 4 AND 5.—*Solid diagrams of deformation temperatures*

proached. From that point the temperature increase with increments of clay is more abrupt.

Decrease in the flux content (feldspar and whiting) increases the temperature of deformation gradually until it approaches 30 per cent, after which further reductions cause a rapid rise in temperature.

The deformation eutectic axis starts at approximately 20 per cent clay, 10 per cent flint, and 70 per cent fluxes, and runs in a straight line to approximately 10 per cent clay, 55 per cent flint, and 35 per cent fluxes. At this point the axis deflects to a higher clay content.

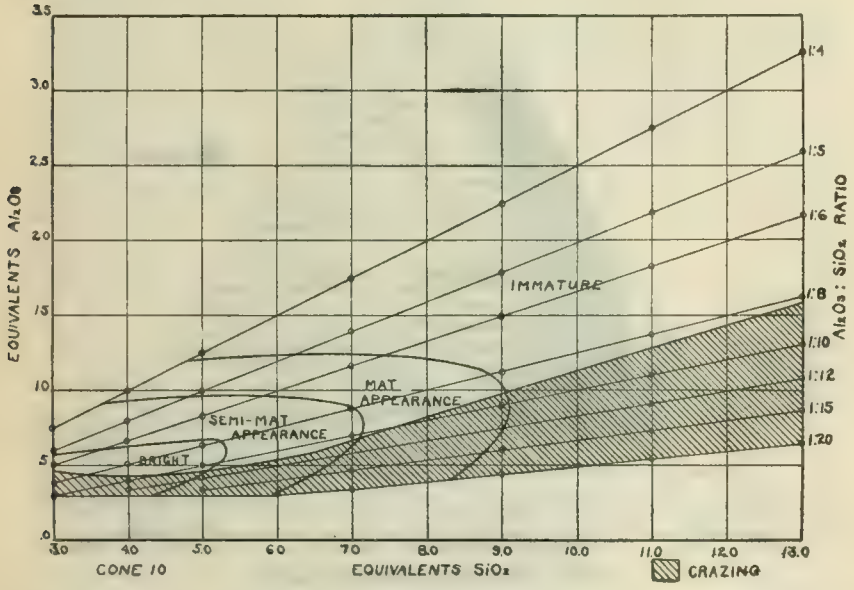


FIG. 6.—*Porcelain glazes at cone 10*

A plaster of Paris solid diagram was made of the field as plotted in Fig. 2 and is shown in Figs. 3, 4, and 5. This shows plainly within what wide limits the more fusible glazes fall.

Cone 10 Burn.—In Figs. 6 and 7 are shown the results of the cone 10 burn. At this heat treatment only a small part of the field is bright and the craze area comparatively large.

It will be noted that there is a belt surrounding the bright area in which the glazes are of semimat appearance, and surrounding this are glazes of mat appearance. The latter have the characteristic mat texture due to high alumina, while for the high silica glazes the term mat appearance is used only as a criterion of

maturity. Outside of the mat area the glazes are dry and immature.

Fig. 6 shows that the obvious way to correct crazing in this type of glaze is to increase the alumina content by addition of clay.

Cone 12 Burn.—At cone 12, Figs. 8 and 9, the area of good bright glazes has increased and the crazing area receded as would be expected. Glazes 2, 3, and 4 show signs of overburning, as

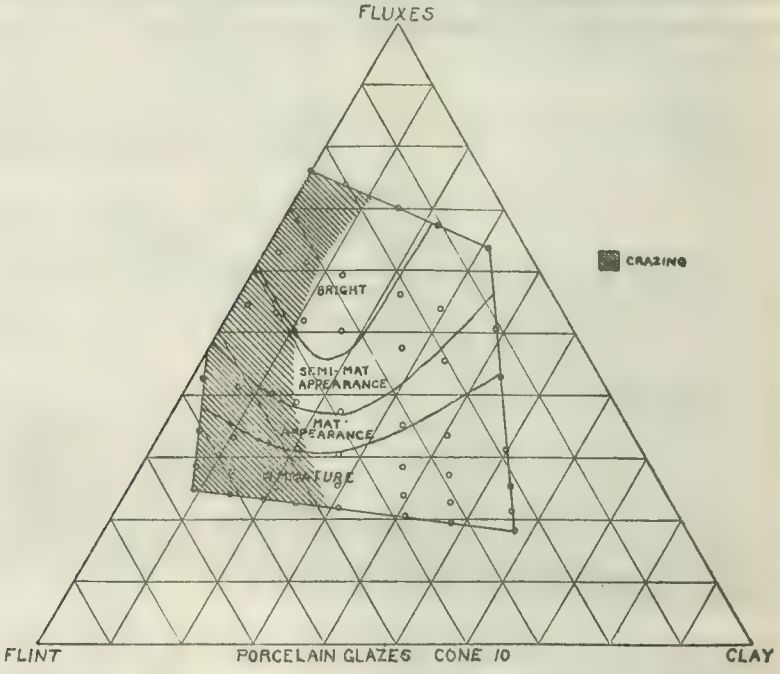


FIG. 7.—*Porcelain glazes at cone 10*

evidenced by the formation of bubbles which were not present at cone 10. The semimat and mat areas have shifted position in accordance with the change in the area of bright glazes.

Glazes C_1, C_2, C_3, C_4, B_3, B_4, and B_5, which are the high silica glazes of lowest alumina content and the only glazes of 55 per cent or more flint, show signs of segregation at this fire. This evidenced itself by lines on the surface of the glaze similar to watermarks, but rising from the surface instead of being depressions. It appeared as if the fluxes were drawn to the inside of the glaze, leaving a skeleton of flint at the surface. With the contraction of the body in burning, these high flint glazes did not follow, and consequently had to rise from the surface of the body.

Cone 14 Burn.—Figs. 10 and 11 show that at cone 14 the area of bright glazes and the area of overburned glazes has increased, the craze area receded still further, and the semimat and mat area shifted again.

The softer of the overburned glazes at this temperature were full of bubbles and had run to the bottom of the piece badly.

It was interesting to note that glazes 2 and 3, which were badly overburned and were far out of the craze area at this heat treatment, were crazed. No. 3 had been sound at cones 10 and 12. This seems to be evidence that when this type of glaze is over-

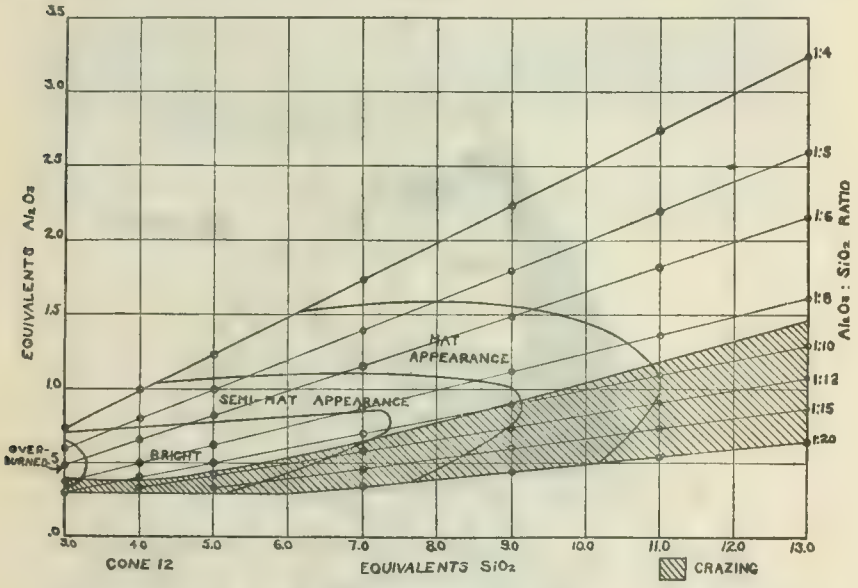


FIG. 8.—*Porcelain glazes at cone 12*

burned sufficiently, its composition or its physical state is changed enough to cause crazing.

The seven glazes containing 55 per cent or more flint showed segregation as at cone 12.

Cone 16 Burn.—At cone 16, Figs. 12 and 13, the area of bright glazes crosses the entire field, the area of overburned glazes is enlarged and the crazing area smaller. The semimat and mat areas have shifted in accordance with the change in the bright area.

The glazes which were overburned at cone 14 were not included in the cone 16 burn, so that further observation of the effect of

overburning on crazing was not available, as none of the glazes which were overburned at cone 16 but matured at cone 14 had been overburned as much as glazes 2 and 3 at cone 14. The segregation is present in the same glazes at cone 16 as at cone 14.

It is to be noted that although a great many of these mat glazes are such due to immaturity as may be seen by following the charts through the successive burns, those mats and semimats in which the alumina: silica ratio is as high as 1:4 for 3–6 equivalents of silica and as low as 1:6 at 10 equivalents of silica (see

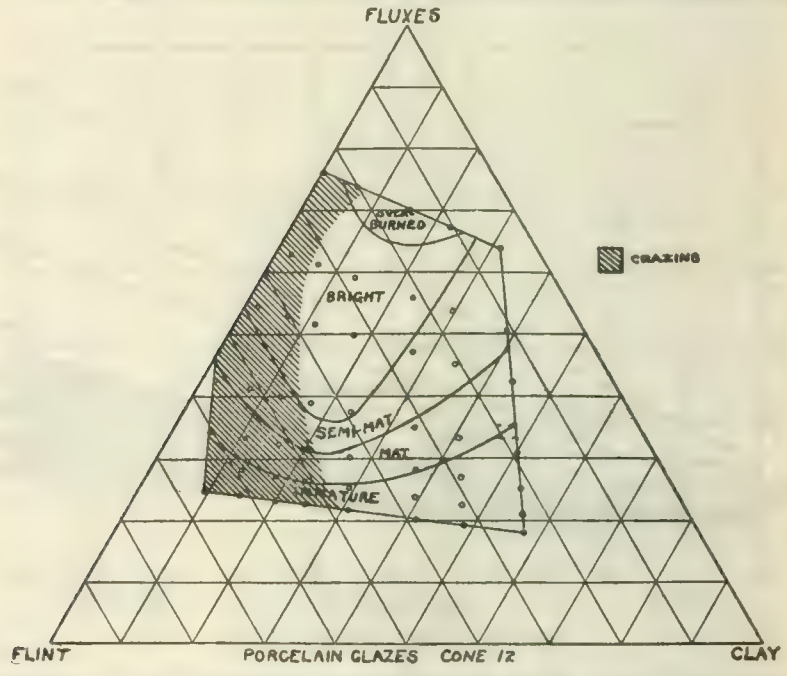


FIG. 9.—*Porcelain glazes at cone 12*

Fig. 12) are not mats of immaturity. Glazes 5, 10, and 15, which have lower deformation points than most of the bright glazes at cone 16, are still semimats. Glaze No. 5, with a deformation temperature of 1134° C, is still a semimat at cone 16.

In Fig. 12 the high gloss axis and the deformation eutectic axis are plotted for comparison. Except in the vicinity of 0.3 equivalents silica, the best glazes of a given silica content are higher in alumina and a trifle harder than the most fusible glaze of that silica content. The eutectic axis lies partially in the craze area.

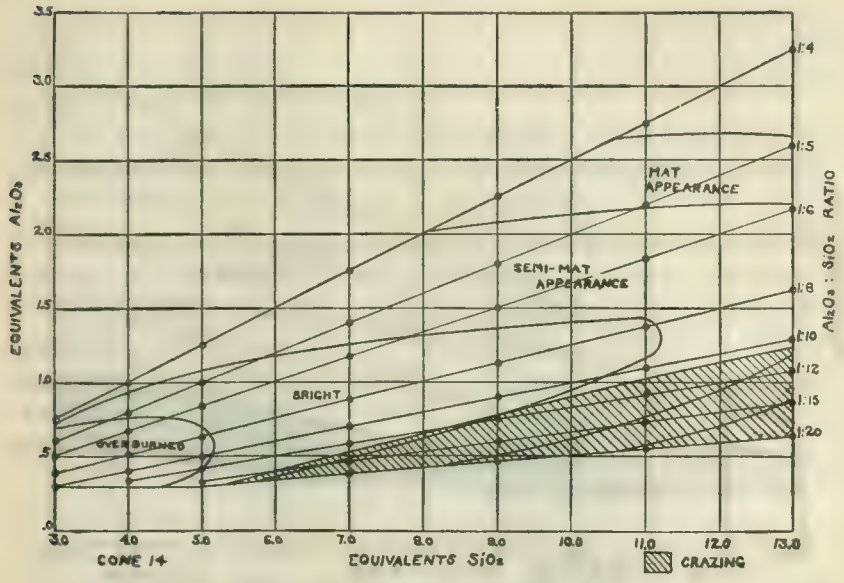


FIG. 10.—*Porcelain glazes at cone 14*

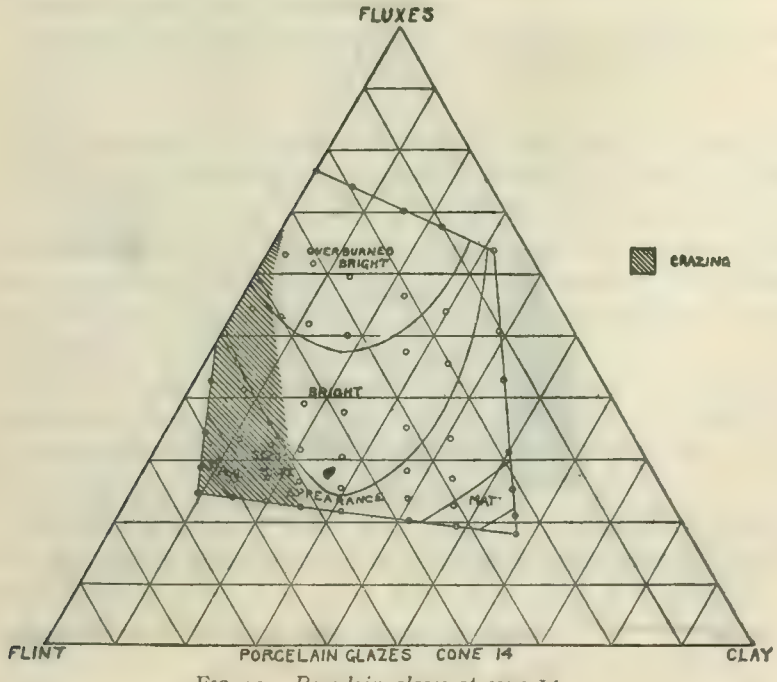


FIG. 11.—*Porcelain glazes at cone 14*

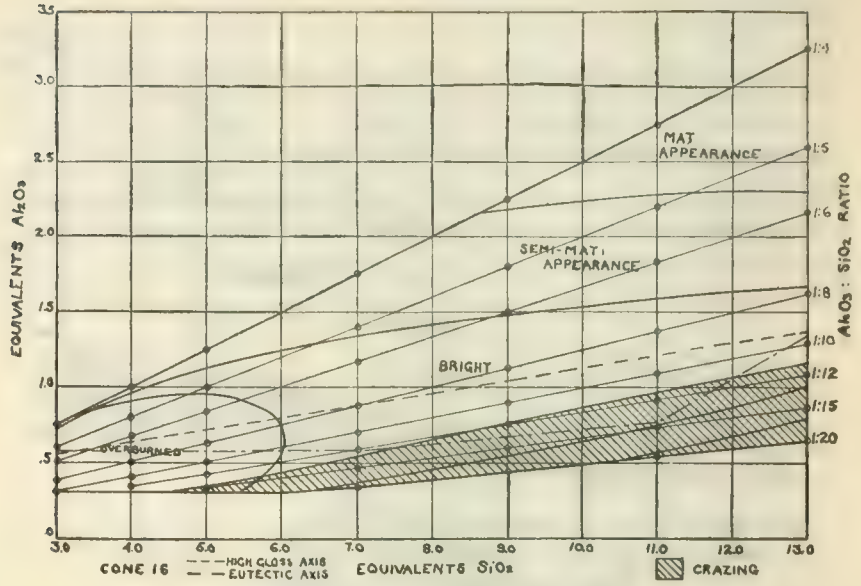


FIG. 12.—*Porcelain glazes at cone 16*

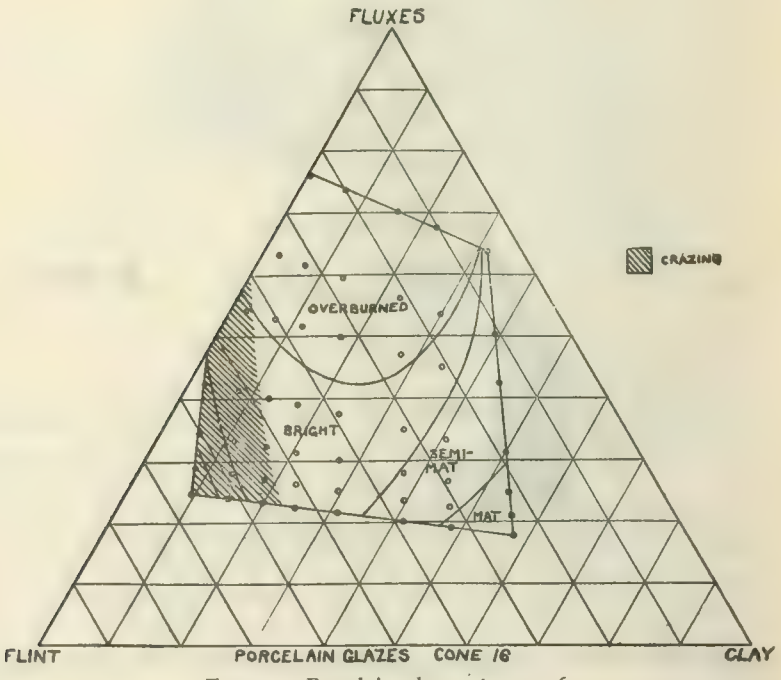


FIG. 13.—*Porcelain glazes at cone 16*

CONCLUSIONS

There is a wide range of commercially acceptable bright glazes of this type maturing from cones 12 to 16 having a temperature range of from four to five cones.

The best glazes lie along a high gloss axis which is practically a straight line represented by the approximate formula

$$Al_2O_3 = 0.3 + 1/12\ SiO_2$$

The range in silica content for a given temperature is wide and becomes greater as the maturing temperature is raised, the possible variations being between 3 to 13 equivalents of SiO_2. The range in alumina content for a given temperature is very much narrower than the range in silica content.

There is a deformation eutectic axis for this series which lies below the high gloss axis and partially in the crazing area.

The best bright glazes for each heat treatment are found at the center of the area of mature good glazes on the chart.

Mat glazes of this type in which the alumina is high enough to give an alumina: silica ratio of 1:4 for 3 to 6 equivalents of silica and tapering down to 1:6 for 10 equivalents silica are aluminous mats and not mats of immaturity since increasing heat treatment will not change them to bright glazes.

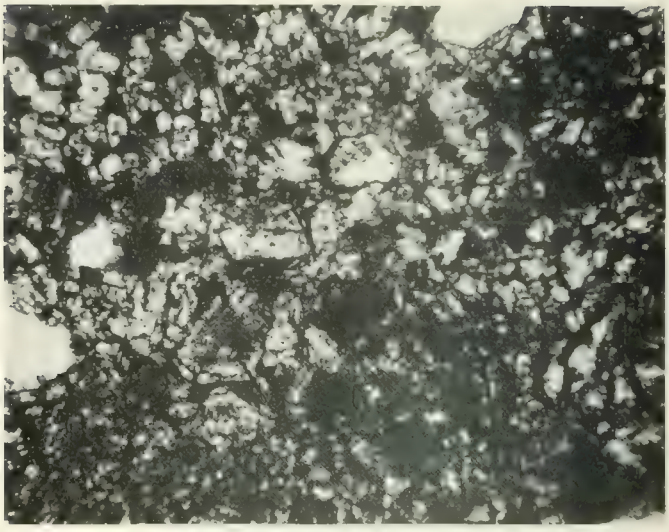
Glazes of this type containing 55 per cent or more flint show segregation and will not lie down to smooth surfaces from cones 12 to 16.

The proper way to correct crazing in this type of glaze is by increasing the alumina through the addition of clay. Higher fire causes the crazing area to recede, but not enough to make higher fire a safe way to correct crazing, except in the glazes containing 4 or less equivalents of silica. There seems to be evidence that this type of glaze will craze when overburned sufficiently, which might be worthy of further investigation.

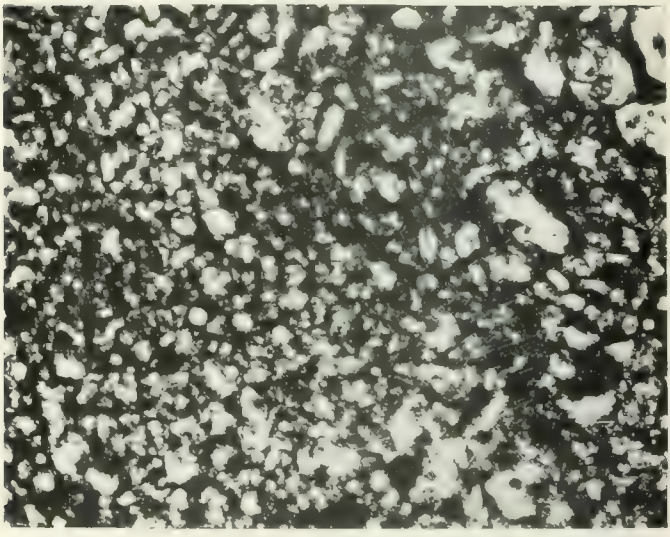
Addition of flint to this type of glaze will not stop crazing, but if carried far enough may throw the composition of the glaze into the crazing area. This is in agreement with Stull, who found Seger's rule did not hold for this type of glaze.

Acknowledgments are due to Dr. A. V. Bleininger for advice and suggestions in this work, and to Robert Sherwood for making the bisque.

WASHINGTON, February 11, 1921.



a



b

FIG. 1.—*Sections of typical Portland cement clinker, showing lack of distinct outline of the constituents.* (*a*) × 115; (*b*) × 135

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CEMENTING QUALITIES OF THE CALCIUM ALUMINATES

BY

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Bureau of Standards

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CEMENTING QUALITIES OF THE CALCIUM ALUMINATES

By P. H. Bates

ABSTRACT

The four calcium aluminates ($3CaO.Al_2O_3$, $5CaO.3Al_2O_3$, $CaO.Al_2O_3$, $3CaO.5Al_2O_3$) have been made in a pure condition and their cementing qualities determined. The first two reacted so energetically with water that too rapid set resulted to make them usable commercially. The last two set more slowly and developed very high strengths at early periods. These two aluminates high in alumina were later made in a pure and impure condition in larger quantities in a rotary kiln and concrete was made from the resulting ground clinker. A 1:1.5:5.5 gravel concrete developed in 24 hours as high strength as a similarly proportioned Portland cement concrete would have developed in 28 days.

Recent investigations [1] have shown that lime can combine with alumina only in the following molecular proportions: $3CaO.Al_2O_3$, $5CaO.3Al_2O_3$, $CaO.Al_2O_3$, $3CaO.5Al_2O_3$. These investigations also show that the tricalcium aluminate is the only aluminate present in Portland cement of normal composition and normal properties. Later these results were confirmed by work carried on at this Bureau and published as Technologic Paper No. 78.[2] In this latter paper some of the properties of the tricalcium aluminate as a cementing material are given. The present paper will present in a brief résumé these same characteristics as well as new information obtained since the first work on this compound was carried out. However, the major portion of this paper will deal with the properties of the other aluminates mentioned above, paying particular attention to the value of these as a cementing material.

Portland cement is composed of from 90 to 95 per cent of lime, silica, and alumina. The ternary system composed of these is therefore of much interest to those concerned with cementing materials, as the composition of Portland cement which will pass standard specifications [3] is confined to rather narrow limits and

[1] Rankin, "The ternary system lime-alumina-silica," Am. J. Sci., 39; January, 1915.

[2] Bates and Klein, Properties of the Calcium Silicates and Calcium Aluminates Occurring in Normal Portland Cement, B. S. Tech. Paper, No. 78.

[3] U. S. Government Specification for Portland Cement, B. S. Circular No. 33. American Society for Testing Materials, Standard Specifications and Tests for Portland Cement, serial designation C 9-17.

covers but a small area in the system when the latter is graphically presented in a triaxial diagram. The question naturally arises, as a consequence, whether any other compositions representing other areas may have cementing properties. In order to determine this point an attempt was made to produce all of the possible compounds in as pure a state as possible. It was not possible to produce all of them of the desired purity and in the quantity necessary to determine their hydraulic properties, on account of the fact that some dissociate at temperatures near their melting points. It was not feasible to prepare these latter in large quantities by crystallizing them from a glass of their composition at a temperature below their melting point. However, the entire ternary system was covered by burning of batches having compositions very close to those of all the different possible compounds, which are:

$CaO.SiO_2$	$3CaO.Al_2O_3$	$Al_2O_3.SiO_2$	$CaO.Al_2O_3.2SiO_2$
$3CaO.SiO_2$	$5CaO.3Al_2O_3$		$2CaO.Al_2O_3.SiO_2$
$2CaO.SiO_2$	$CaO.Al_2O_3$		$3CaO.Al_2O_3.SiO_2$
	$3CaO.5Al_2O_3$		

Some of these exist in two or more forms.

When the burned clinker was ground and gaged with water, it was found that of the above compounds only the α and β forms of the $2CaO.SiO_2$, $3CaO.SiO_2$, and all of the calcium aluminates had such properties as might justify their being called hydraulic cements. However, it is questionable if the two aluminates higher in lime should rightly be considered as having hydraulic properties, as their reaction with water is so energetic and accompanied by such an evolution of heat that a large excess of water is required in gaging. This produces an open porous mass which reduces its resistance to the further action of water if placed therein. They therefore show a decrease in strength when aged under these conditions. Consequently, as hydraulic cements should not show decreasing strength or slow disintegration under such conditions, the $5CaO.3Al_2O_3$ and more particularly the $3CaO.Al_2O_3$, can not properly be called hydraulic cementing materials. It was not thought advisable to make large quantities of these two aluminates on account of these properties. However, those high in alumina—$CaO.Al_2O_3$ and $3CaO.5Al_2O_3$—showed such surprising strength, especially at early ages, that it was thought desirable to make them in larger quantities and also with varying amounts of the three impurities—silica, iron oxide, and magnesia—which generally accompany lime and alumina. When the larger quan-

tities of cement were desired, they were burned in a 2 by 20 foot rotary kiln, the whole process of manufacture in general being similar to that used in producing Portland cement.

The data herewith presented are not the first to be obtained with compounds of this type. Spackman [4] has contributed considerably to the subject, but published nothing showing what compounds were actually present in the cements he used. He furthermore made no concrete test pieces. A rather extended discussion of the aluminates took place following a paper by Killig,[5] at the 1913 meeting of the German Portland Cement Manufacturers' Association. No further work was apparently done until this presented herewith, although, according to Conwell,[6] the La Farge cement works in France produced a high calcium aluminate cement ("artillery cement") for gun foundations during the late war. Endell [7] presented a paper before the 1919 meeting of the German Portland Cement Manufacturers' Association giving results obtained with cements having compositions very similar to the impure aluminates used in this investigation. He also determined the hydraulic properties of all the compounds in the $CaO.Al_2O_3.SiO_2$ system, and concludes that only the orthosilicate of lime, the monaluminate of lime, and the 3:5 calcium aluminate form true cements.

Tricalcium Aluminate $3CaO.Al_2O_3$.—When lime and alumina in the molecular proportions of three of the former to one of the latter are heated together at a temperature much beyond 1350° C, there is not produced the compound $3CaO.Al_2O_3$, but a mixture of $5CaO.3Al_2O_3$ and CaO. The presence of the latter can be readily shown by the petrographic microscope, but not by the physical properties of the ground burned material, as these are very similar to those of ground $3CaO.Al_2O_3$. It may be stated that if the burned mass shows signs of fusion or of marked vitrification then uncombined lime is unquestionably present. This inability of the lime and alumina to combine in the ratio of 3 to 1 at high temperature is due to the fact that such a compound is unstable at its melting point (1535° C), decomposing into $5CaO.3Al_2O_3$ and CaO; this decomposition starts at a much lower temperature than the melting point. To secure the tricalcium aluminate, therefore, the

[4] Spackman, "Aluminates: their properties and possibilities in cement manufacture," Proc. Am. Soc. for Test. Mat., 10, p. 315; 1910.

[5] Killig, "Calcium aluminates and their influence on hydraulic cements," Prot. Vereins Deut. Port. Cement Fabrikanten, p. 408; 1913.

[6] Conwell, Discussion of a paper by Bates—"Cements producing such hardening concretes," Proc. Amer. Soc. for Test. Mat., 19, pt. 2, p. 440; 1919.

[7] Endell, "Cement rich in alumina," Zement, 8, pp. 319–321, 334–336, 347–350; 1919.

practice at this laboratory has been to heat the mixture of the proper composition to 1400 to 1450° C for about half an hour, cool, grind, and reburn at a temperature of 1300 to 1350° C for several hours, cool, and examine microscopically for free lime. If it is present, the burned material is reground and reburned at the lower temperature mentioned until the microscope no longer reveals free lime.

The addition of water to this freshly prepared aluminate produces a very vigorous reaction, manifesting itself almost immediately by a rapid rise in temperature until the mass steams vigorously and assumes a "flash set." This is a partial hydration and is confined largely to the exterior of the individual grains or agglomerations of these. If the kneading with water is continued after the mass has cooled somewhat, there is produced a very plastic mass. It is necessary to use initially from 40 to 45 per cent of water in order to have enough present after the steaming to permit of the continued kneading. When this action is carried on vigorously for several minutes, there is produced a smooth plastic mass, which, with still further working but with no further addition of water, becomes so fluid that it will not permit of the making of a pat. It has not been possible to produce a smooth pat free from large agglomerations of unaffected aluminate by the use of less water than the amounts given above.

The partially hydrated aluminate, after the continued kneading following the flash setting, acquires a kind of final set. This may take place in an hour or may require two or three days, depending upon how hot the mass is after the kneading. If it still remains very warm, the heat seems to accelerate the setting, especially by evaporation. On the other hand, if it has become cooled to about room temperature, a much longer period ensues before setting takes place.

The test pieces after final setting has taken place are still comparatively soft and can be indented with the finger nail. Any surface not exposed to the atmosphere during setting is perceptibly softer than one which has been so exposed. If the kneading following the flash set has not been thorough enough to completely break up all agglomerations and make a smooth paste, the "steaming test" will disintegrate the test pieces.

The effects of the addition of SO_3 in the form of plaster of Paris to the aluminate alone and to the latter when it contained 2 per cent of hydrated lime are shown in Table 1.

TABLE 1.—Time of Set of Tricalcium Aluminate

[All samples gave a flash set. All results shown were obtained by thorough kneading of the set material]

Percentage composition				Penetration, Vicat plunger	Initial set		Final set	
$3CaOAl_2O_3$	Gypsum	$Ca(OH)_2$	H_2O		Vicat	Gilmore	Vicat	Gilmore
					H. m.	H. m.	H. m.	H. m.
100			40	None	(a)	(a)	(a)	(a)
100			45	do	(a)	(a)	(a)	(a)
97.5	2.5		41	do	(a)	(a)	(a)	(a)
97.5	2.5		45	do	(a)	(a)	(a)	(a)
95.5	2.5	2.0	41	do	(a)	(a)	(a)	(a)
95.0	5.0		41	9 mm	0 07	0 14	5 20	(b)
93.0	5.0	2.0	41	To bottom	0 23	0 43	1 30	4 30
93.0	5.0	2.0	37	11 mm	0 21	0 38	1 15	1 15
90.0	10.0		41	do	0 23	0 52	0 50	1 00
88.0	10.0	2.0	41	To bottom	0 30	1 15	0 55	1 20
88.0	10.0	2.0	37	15 mm	0 25	1 10	0 55	1 15
85.0	15.0		41	To bottom	1 10	3 50	6 30	6 30
85.0	15.0		37	15 mm	0 40	3 00	6 30	6 30
83.0	15.0	2.0	37	20 mm	0 25	1 50	6 00	7 30
83.0	15.0	2.0	34	6 mm	0 18	1 45	(b)	(b)
80.0	20.0		41	To bottom	1 00	3 50	(b)	(b)
80.0	20.0		37	15 mm	0 45	3 50	(b)	(b)
78.0	20.0		37	25 mm	0 55	3 50	(b)	(b)
78.0	20.0	2.0	34	7 mm	0 50	3 50	(b)	(b)

[a] Violent steaming prevented continuous kneading. (See text.)
[b] Set between 7½ and 22 hours.

It is seen from the table that 2 per cent of hydrated lime, or 2½ per cent of plaster alone or in the presence of 2 per cent of the hydrate, changed the setting properties but little. There resulted in all cases a flash set followed by a violent steaming which necessitated the stopping of the kneading for a time. Later the mass could be kneaded into a smooth plastic condition which, as stated above, gave a final set of from 1 hour to 2 or 3 days. Five per cent of plaster retarded the steaming sufficiently to permit of the making of test pieces. These gave off steam at the time of initial set, which varied considerably with the Vicat or Gilmore test pieces, on account of the different volumes of the test pieces. Five per cent of plaster in the presence of 2 per cent of hydrated lime produced a much more marked retardation of the initial set, but accompanied as before by an evolution of steam. Increasing amounts of plaster produced increased retardation of the initial and usually also of the final set. Steaming occurred at about the time of final set when 10 or 15 per cent plaster was used. Lime hydrate in the presence of these larger amounts of gypsum decreased the amounts of water required for gaging. It should be particularly emphasized that flash setting occurred in all cases, and the results given were obtained by thorough kneading after the appearance of this phenomenon.

The Vicat test pieces placed in water at the end of 24 hours disintegrated in the course of a few hours. If allowed to harden for several days they did not disintegrate rapidly, but in the course of several months they checked on the surface and increased very much in volume.

On account of these several properties and the fact that previous work of the Bureau showed that this aluminate did nor develop any strength as a bonding material, no specimens to determine the strength were made in this investigation. The tricalcium aluminate is apparently not a true hydraulic cement, and the heat evolved in gaging with water and the later increase in volume do not permit of its wide practical application.

The question may arise as to why Portland cement, which contains approximately 20 per cent of this compound, does not show more of the characteristics of the latter. The answer is to be sought perhaps in the condition in which the tricalcium aluminate occurs in the cement clinker as revealed by examining through the microscope a thin section of clinker. It is then seen that in normal clinker the constituents do not exist with a well-defined outline nor separated into well-defined crystals. This is shown in Fig. 1, photomicrographs made by transmitting light through a thin section of a typical Portland cement clinker. The dicalcium silicate, the major constituent, exists in fairly large grains which blend off into a more or less homogeneous boundary containing the tricalcium silicate, tricalcium aluminate, and the dark-colored iron compounds. Although in a clinker burned at a high temperature for a comparatively long time and having a high silicate content, the tricalcium silicate may exist in large, well-defined crystals, such a clinker is an exceptional one. It can be readily seen that in grinding normal clinker, the grains will not be separated grains of any one constituent, as they would be in a mechanical mixture, but even the finest ground particles may be composed of all the constituents. Therefore the activity will be reduced a certain degree by the inability of the water to reach all of its surface.

The activity of the aluminate will be furthermore masked by the presence of the dicalcium silicate which is present in amounts about two and a half times as great. This is relatively very inactive toward water, shows a setting time of several days, and does not evolve heat to a perceptible degree. The activity is furthermore modified by the presence of the gypsum added during the grinding of the clinker. This latter is added usually to an

amount equal to 3 per cent of the weight of the clinker, which is equivalent to about 15 per cent of the aluminate. Also on adding water to ground cement, the tricalcium silicate starts to hydrate by splitting off lime. The effects of both lime and gypsum on reducing the activity of the pure aluminate have been discussed above, and these in connection with the condition in which the aluminate exists in clinker explain why its properties when in the pure condition are not distinctly manifested when in Portland cement clinker.

$5CaO.3Al_2O_3$.—This aluminate can be prepared in the pure condition without any great difficulty, as it shows no tendency to decompose at temperatures approaching its melting point. Its activity toward water is distinctly less than that shown by the tricalcium aluminate. By the use of about 10 per cent more water (in terms of the cement) than is used in gaging Portland cement it is possible to knead thoroughly the aluminate with water without the appearance of any flash set. Initial set occurs in from 3 to 5 minutes after molding, and final set occurs in from 15 to 30 minutes, the latter being accompanied by a marked evolution of heat. Plaster of Paris retarded the initial set to a very slight degree, but the final set was retarded about one hour.

This compound attained considerable strength both in the neat condition and in a 1:3 mortar. Thus the average neat tensile strength of two burnings at 7 days was 275 pounds per square inch, and at 28 days 311 pounds; the neat compressive strength was 3090 pounds and 3670 pounds at 7 and 28 days, respectively. The tensile strength of the 1:3 standard sand mortars was relatively much higher, being 261 pounds per square inch at 7 days and 264 pounds at 28 days. The mortar strength was also high in compression, being 1750 pounds at 7 days and 1300 pounds at 28 days. The addition of plaster produced a 50 to 100 per cent increase in strength. This was more marked at 28 days, so that the slight increases in strength (or retrogressions in some cases) between the 7 and 28 day periods were very much enlarged.

The hydration of this aluminate consists in the formation of hydrated tricalcium aluminate and hydrated alumina. The hydration of all of the aluminates consists of this same reaction, all of the alumina in excess of that required for the ratio of three lime to one of alumina forming a gelatinous mass of hydrated alumina when the aluminate is completely hydrated. As this is the molecular ratio of the components of tricalcium aluminate, of necessity no alumina is split off from the latter and the sole

product of its hydration is tricalcium aluminate having a varying number of molecules of water present, depending upon whether it exists in a colloidal or crystalline form.[8]

This aluminate was not produced in large enough amounts to permit of the making of concrete specimens on account of its rapid setting qualities. Simultaneous work with the two other aluminates of higher alumina content showed that they did not possess this property of rapid set but did develop very high strengths. Therefore further work on the $5CaO.3Al_2O_3$ compound was discontinued, and the remainder of the investigation was devoted to $CaO.Al_2O_3$ and $3CaO.5Al_2O_3$, both in the pure and impure condition.

$CaO.Al_2O_3$ and $3CaO.5Al_2O_3$.—As the preliminary work with these two aluminates showed that they had such properties as would recommend them for use as hydraulic cements, it was thought desirable to make burnings of such a size as would give sufficient material to permit of making concrete specimens to be broken at late ages as well as the usual small mortar test pieces. Eight burns in all were made in the 2 by 20-foot rotary cement kiln of this Bureau. In these the alumina content varied from 30.5 to 74.1 per cent. The constituents, other than lime and alumina, in three of the burns were as low as it was possible to have them without using chemically pure lime and alumina. In these three particular burns, where the desire was to produce either $CaO.Al_2O_3$ or $3CaO.5Al_2O_3$ in as pure condition as possible, the total amount of silica, iron oxide, and magnesia did not exceed 1.89 per cent in the clinker. Varying amounts of these three impurities were introduced purposely in the other five burns to determine how the properties of the pure aluminate would be affected by these. The minimum amount of silica was 9.41 per cent, the maximum amount 17.38 per cent; the minimum amount of iron oxide was 1.85 per cent and the maximum 3.10 per cent; and the magnesia had 1.04 and 3.66 per cent as the minimum and maximum percentages. These limits were used because it was believed that they would represent the amounts of impurities that might be encountered in using commercially available sources of raw material.

The pure aluminates were made from a limestone low in all impurities, and calcined alumina. The former came from a large deposit of a very pure limestone in York County, Pa., and the latter was purchased from a dealer in certain ceramic supplies and

[8] Klein and Phillips, Hydration of Portland Cement, B. S. Tech. Paper No. 43.

was made from either bauxite or cryolite. While limestones comparatively low in impurities are widely scattered throughout the country, such is not the case with respect to alumina. In this case, were it desired to produce the lime aluminate on a commercial scale, recourse to bauxite or to highly aluminous clays as a scource would be necessary. These would be used to the maximum allowable amount, as determined by the effects of the impurities present, and any deficiency in alumina would be made up by adding such a material as the above-mentioned calcined alumina, or an alumina obtained as a by-product in some of the recently established processes of obtaining potash from alunite or felspar. The high cost of a pure alumina would totally preclude its use as the sole source of the alumina in the aluminate.

The impure aluminates referred to in this investigation (see Table 2) as Nos. 3, 4, and 5 were made of a raw batch composed of the above-mentioned limestone, kaolin, alumina, and iron oxide. The raw batch of No. 6 contained in addition to these some flint, while that of No. 7 contained all of these but the kaolin, which was partially replaced by bauxite.

The composition of the raw batches, the analyses of the clinker, and the temperature at which the burnings were made are given in Table 2. The raw batch was ground to such a fineness that at least 85 per cent passed a 200-mesh sieve. The alumina and flint, as received, were of a fineness of more than 90 per cent through the same sieve. The temperatures of burning given in the table are the average of temperature readings taken every 15 minutes throughout the burns, which usually required from 4 to 5 hours. The pyrometer used was of the Holborn-Kurlbaum type, and the readings are probably accurate to $\pm 15°$.

TABLE 2.—The Raw Composition, Analyses, and Temperatures of Burning of the Clinker

Cement No.	1	2	3	4	5	6	7	8
Raw composition:								
Limestone.....pounds..	350.0	350.0	280.0	341.5	315.0	396.0	325.0	255.0
Alumina.....do....	360.5	360.5	238.0	172.5	108.0	94.5	190.0	450.0
Kaolin.....do....	.....	.....	105.0	97.5	180.0	93.0	.....	.....
Iron oxide.....do....	.....	.....	10.5	9.5	9.0	5.0	5.0	.....
Bauxite.....do....	.....	.....	.....	.....	.....	.....	160.0	.....
Potters flint.....do....	.....	.....	.....	.....	.....	37.5	.....	.....
Clinker analyses:								
SiO_2.....	0.44	0.76	9.41	10.48	17.23	17.38	11.33	.68
Al_2O_3.....	62.31	61.25	55.09	46.71	39.96	30.52	47.66	74.11
Fe_2O_3.....	.51	.60	2.04	2.13	2.57	1.85	3.10	.40
CaO.....	36.69	36.32	30.73	39.79	38.84	46.72	34.87	23.82
MgO.....	.36	.48	2.95	1.04	1.29	2.27	3.66	.81
Ig. loss.....	.07	.50	.08	.32	.14	.78	.17	.38
	100.88	99.91	100.30	100.47	100.03	100.52	100.19	100.20
Temperature of burning, degrees centigrade.....	1460	1480	1490	1380	1455	1360	1445	1500

When this investigation was started, the work of Rankin on the lime-silica-alumina system had not been published, and consequently the temperatures which any burn would require were known only in a very general way. However, in all cases the burnings were made at the lowest temperature at which the clinker would appear to show the start of vitrification. When such clinker appeared the temperature was allowed to vary as little as possible. It must be stated, however, that generally the impure aluminates were burned at too high a temperature, and the clinker was harder than desired. This was due to the fact that these exhibited a very narrow range of temperature between that at which they were underburned (showing a tendency to "dust") and that at which they formed "balls" or "logs" or even melted in the kiln. This is true of burns Nos. 5 and 6, and especially true of the latter, which unless quenched in water on falling from the kiln dusted rapidly and completely. Quenching, however, was not followed, but the clinker was allowed to cool in the air.

It can be noted from the data that the temperatures of burning are not much higher than those used in the production of Portland cement. Seventy-five samples of "raw mixes" received from various manufacturers of Portland cement in the United States gave an average clinkering temperature of 1435° C. This was the average lowest temperature at which the clinker from these raw mixes showed microscopically the absence of free lime. The time of burning of these was the same, as stated by the manufacturers furnishing them, as that required to produce the cement commercially. Furthermore, while 1435° C was the average temperature of burning of the commercial raw mixes, yet some required a temperature in excess of 1500° C. The temperatures used in producing these aluminates are not therefore in excess of those used in commercial rotary kiln practice.

One point in connection with the burning should be particularly emphasized, and that relates to the nature of any underburned products. Underburning to a certain degree in the case of Portland cement produces what is called "unsound" cement. This is cement which, when gaged with water and the hardening accelerated at the end of 24 hours by heating at 100° C in an atmosphere of steam, shows such permanent expansion that the test piece warps, cracks, or softens. This expansion is due especially to the presence of too much uncombined lime.[9] In these

[9] It is not within the province of this paper to enter into a detailed discussion of all the phases of the "unsoundness" of Portland cement or how it is fully caused. The above statement covers in a broad but brief way the cause and manifestation of these phenomena, but does not go into the details of showing how a certain amount of uncombined lime in one clinker may exhibit unsoundness and the same amount in another not show this condition.

aluminates, particularly Nos. 1, 2, and 8, the total amount of lime is very low, when compared with the amount in Portland cement, and the alumina correspondingly high. The lime will have all combined during the burning, and any uncombined material will be alumina. This latter does not react with water, as does lime, and therefore will not show the expansion and disruption shown by the free lime of Portland cement. Underburning of the impure aluminates is shown particularly by the dusting of the clinker. This is due to the fact that the burning has not been conducted at a sufficiently high temperature (or for a sufficiently long period at a lower temperature) to produce the β form of the orthosilicate in a stable state. In cooling it reverts to the γ form of the same silicate accompanied by an increase in volume of about 10 per cent. Even in the impure aluminates, the relative amounts of lime are so low that at low temperatures it is all combined with silica or alumina and no products are present which manifest excessive expansion on hydrating. Consequently it is not possible to produce unsound cement with materials approaching the composition used in this investigation without the use of temperatures far lower than those given in Table 2.

The addition of the impurities tended to cause a lowering of the temperatures required for satisfactory burning. This is in accordance with the rule that generally the introduction of a third compound into a binary system produces a lowering of softening points as the ternary field is entered. Cement No. 3 is no exception to this statement, notwithstanding the fact that it was burned at an average temperature of 10° C. higher than No. 2. The clinker produced in the former case was decidedly harder than the latter. To those engaged in the manufacture of Portland cement, the fact that an increase of lime produces a lowering of clinkering temperature may seem strange. But the work of Rankin, previously cited, shows that the trend of the softening points of lime-alumina mixtures is downward from the melting point of pure alumina at 2050° C. to the melting point of the eutectic between $3CaO.Al_2O_3$ and $5CaO.3Al_2O_3$ (50 per cent lime and 50 per cent alumina) at 1395° C, and then upward to the melting point of pure lime at 2570° C. The compositions of the aluminates here considered all lie between $5CaO.3Al_2O_3$ and Al_2O_3, hence an increase in lime will produce a lowering of the clinkering temperatures except in the neighborhood of eutectic points. These latter, however, are characterized in the binary system by a slight temperature change for marked changes in composition,

and consequently would not be very pronounced in considering trends of temperature changes. Fig. 2, which shows the location of the compositions under investigation in the ternary system, also shows the melting temperatures of the compounds of lime and alumina and their eutectics according to Rankin. This should materially assist in understanding the changes in clinkering temperatures produced by changes in composition. This applies only to compositions containing lime, silica, and alumina, but as stated

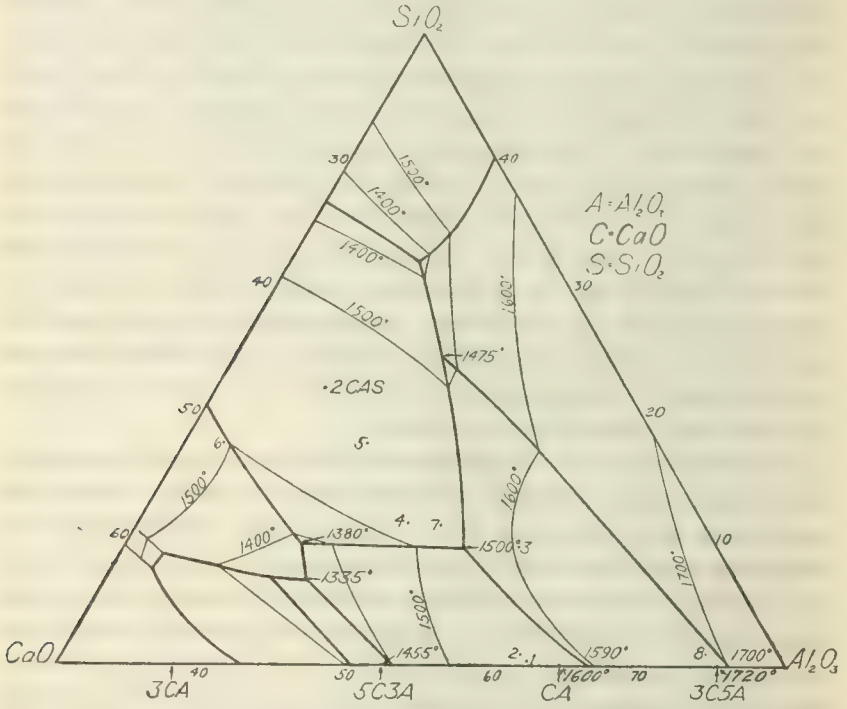


FIG. 2.—*Projection of a portion of the ternary system $CaO.Al_2O_3.SiO_2$ showing the location of the cements under investigation, with isotherms (light lines), boundaries (heavy lines) between some of the fields, and melting temperatures of some of the compounds*

before, the introduction of impurities tends to lower the clinkering temperatures.

The appearance of the clinker of the several burns was of course decidedly different from that of Portland cement. That from Nos. 1, 2, and 8 was whiter than the clinker of white Portland cement and was composed of soft nodules about 1 inch or less in diameter. No. 6 gave a gray slaglike clinker which rapidly dusted to a light-gray dust. No. 5 also was of a gray color, but the exterior of the nodules was rough like Portland-cement clinker,

while the interior was more completely vitrified. Dusting of this clinker was pronounced but not excessive. No. 4 gave a yellow slaglike clinker. Nos. 3 and 7 gave a hard clinker of about the size obtained with Portland cement, black and well vitrified on the interior, but yellow to reddish brown on the exterior. The black glistening appearance of the exterior of Portland cement clinker was at all times lacking.

All clinker was examined petrographically to determine the character and relative amounts of the constituents. These data are given in Table 3. Here the constituents are arranged in order of the amounts present in each particular case. This table should, however, be studied in connection with Table 2 showing the chemical analyses, otherwise it is likely that the proper conception will not be formed of the relative amounts of some of the constituents, as the orthosilicate or the ternary compounds, which are present. Thus while Nos. 3 and 8 show the same constituents in the same order, manifestly No. 3 must contain much more of the ternary compound than No. 8, as the latter contains but 0.68 per cent silica while the former contains 9.41 per cent, and, furthermore, in both cases all of the silica is combined as $2CaO.Al_2O_3.SiO_2$.

TABLE 3.—The Constituents of the Clinker

[C=CaO; A=Al_2O_3; S=S_iO_2]

Cement No.....	1	2	3	4	5	6	7	8
Major constituent.....	3C5A	CA	CA	2CAS	2CAS	2CS	2CAS	CA
	CA	5C3A	3C5A	CA	CA	5C3A	CA	3C5A
Minor constituent.....	2CS	2CS	2CAS	2CS	2CS	CA	2CS	2CAS

The clinker of Nos. 1 and 8 was soft and very fine grained. The petrographic examination of the former was especially difficult, but judging largely from the indices of refraction it was composed mainly of $CaO.Al_2O_3$. This same compound was rather well developed in No. 8, but the main constituent in this case was $3CaO.5Al_2O_3$ existing in poorly defined grains. No. 3 gave a clinker with much better defined crystals, and the constituents noted could be identified comparatively readily. The other burns yielded clinker in which some of the constituents were very well crystallized. These are shown in Fig. 3. These crystalline compounds are either $3CaO.5Al_2O_3$ or the ternary compound $2CaO.Al_2O_3.SiO_2$. Their optical properties are such that they can not be satisfactorily differentiated in thin sections. However, the

immersion method of examination showed that Nos. 5 and 7 contained no $3CaO.5Al_2O_3$. Hence in these two cases the crystals are those of the ternary compound. These sections are typical of the burns containing the impurities. However, the crystalline portions do not extend throughout the mass, but are segregated in certain portions. In this respect they resemble crystalline Portland-cement clinker where crystals in even well crystallized clinker occupy but a comparatively small percentage of the mass and are invariably localized.

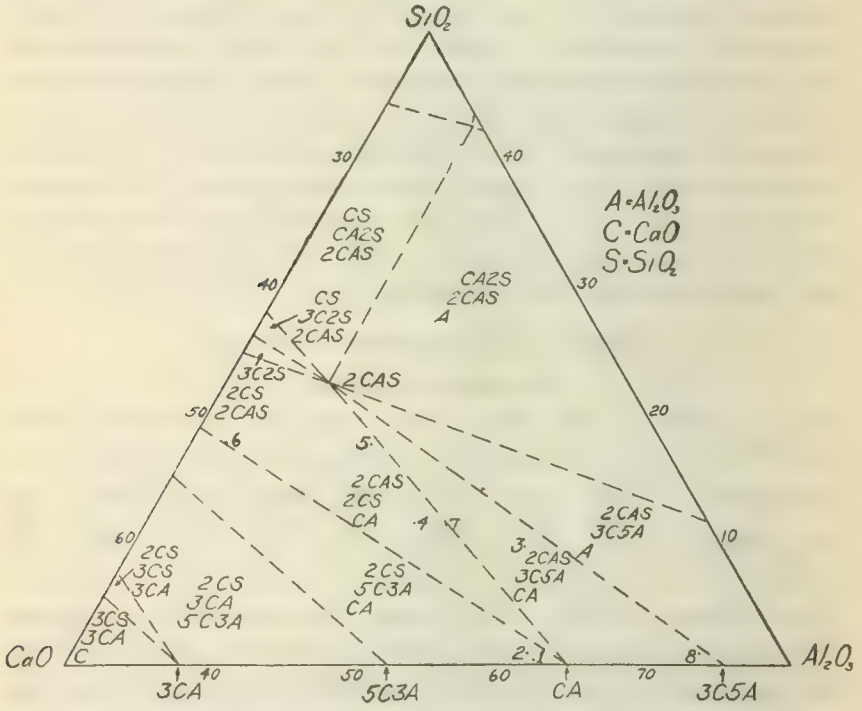
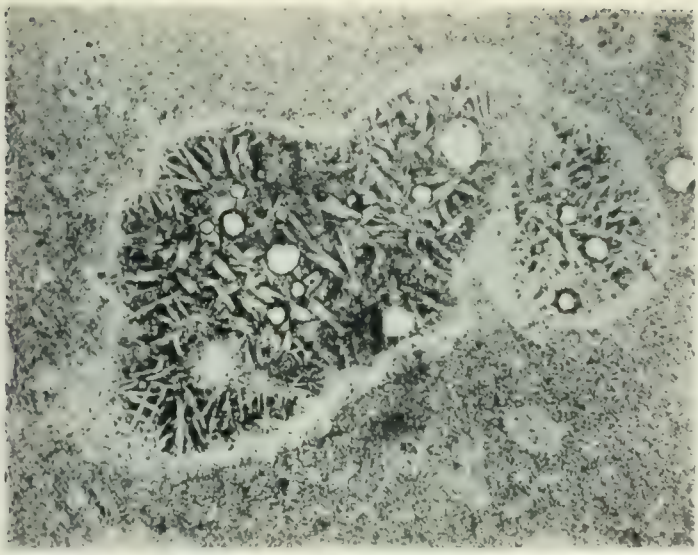


FIG. 4.—*Showing the location of the cements under investigation in the ternary system and the final products of crystallization of solutions of* $Al_2O_3.SiO_2.CaO$

There is shown in Fig. 4 the position of these cements in the ternary system. (In locating these the iron oxide has been considered as alumina and the magnesia as lime.) This figure shows but that portion of the entire system which concerns this investigation. The special reason for its presentation here is to show what the final products of crystallization would be according to Rankin, and to enable one to compare them with the products actually found as shown in Table 3. Any composition of the oxides which lies within any of the areas partly closed by dotted lines would, when equilibrium has been reached, contain either



a

b

FIG. 5. *Examples of crystallization in the aluminate cements; (a) from a thin section of cement 3, × 50; (b) from a thin section of cement 5, × 90*

those compounds indicated as occurring within these areas or else a compound indicated as occurring in one of the adjoining areas. These areas are furthermore triangles (as would be seen if the whole ternary system were reproduced) and are formed by lines drawn from points representing compositions of either binary or ternary compounds. The compounds indicated as appearing in any triangle are those whose compositions are located at one of the apices.

A comparison such as indicated above reveals that notwithstanding the fact that the burning at the recorded temperature was but for a comparatively brief period, yet the constituents present were those which would result if the burning had continued until equilibrium would have been reached. Attention should be called to the fact that the petrographic examinations of the clinker were made before the work of Rankin was at hand, and hence it is very gratifying to note how the work of the latter corroborated the above examinations.

The clinker, without any storage, was crushed and ground to the fineness shown in Table 4. After crushing, but before grinding in ball mills, the clinker was divided into two portions. The one portion was ground with 3 per cent of plaster of Paris, the other was ground with no addition of foreign materials. It was soon found that the cements containing the plaster were not developing as high a strength in concrete as those free of this material. Consequently the concrete specimens in the later burns were made mostly from unplastered cements. In Table 4 and in the remainder of this paper the cements containing the plaster are referred to by the proper numeral and the letter "P"; when referring to the unplastered cements, the numeral alone is used.

TABLE 4.—Fineness of the Cement

Cement No.	Per cent passing—		Cement No.	Per cent passing—	
	100-mesh sieve	200-mesh sieve		100-mesh sieve	200 mesh sieve
1	98.8	83.2	4P[a]	97.0	78.0
1P[a]	99.2	82.8	5	97.6	80.2
2	98.0	80.0	5P[a]	97.8	79.4
2P[a]	99.6	82.2	6	93.0	76.4
3	99.0	81.2	7	98.2	79.8
3P[a]	98.8	81.8	8	99.7	86.6
4	96.8	78.0			

[a] Indicates the addition of plaster.

In Table 5 is presented the time of set of both the plastered and the unplastered cements as determined by both the Gilmore and Vicat needles. The setting time is either too rapid or too slow when compared with the time of set required for Portland cements of standard specification. These latter require an initial set of not less than 60 minutes when the Gilmore needle is used, not less than 45 minutes when the Vicat needle is used, and a final set within 10 hours as determined by either needle. But while too early an initial set may interfere in the practical use of a cement, nothing can be said against a final set in excess of 10 hours if the 24-hour strengths are high enough. Such is the case with the three slow-setting cements of this group. While they have not attained a final set within 8 hours, they have in all three cases in 24 hours a tensile strength as a 1:3 standard sand mortar in excess of that required of Portland in 7 days, and in two of the three cases a strength in excess of that required at 28 days.

TABLE 5.—Consistency and Time of Initial and Final Sets

Cement No.	Initial		Final		Consistency (per cent water)
	Gilmore	Vicat	Gilmore	Vicat	
	H. m.	H. m.	H. m.	H. m.	
1	0 55	0 25	4 45	2 10	27.0
1P	0 50	0 15	4 55	2 30	27.0
2	0 30	0 10	2 00	0 50	25.0
2P	0 40	0 05	3 50	0 50	25.0
3	6 25	5 15	(a) (a)	7 05	21.0
3P	0 25	0 20	6 20	3 25	21.5
4	0 35	0 10	3 30	1 15	22.0
4P	0 10	0 03	1 40	0 25	20.0
5	0 40	0 15	4 45	3 15	21.5
5P	0 15	0 15	3 00	0 30	21.5
6	0 07	0 02	0 45	0 12	29.0
6P	Flash	Flash	Flash	Flash	30.0
7	6 00	3 45	(a) (a)	(a) (a)	19.0
7P	0 35	0 20	4 30	2 30	19.0
8	7 00	3 30	(a) (a)	(a) (a)	26.0
8P	1 05	0 20	4 00	2 45	26.0

[a] Final set occurred between 8 and 16 hours after mixing.

When the time of set is studied in connection with the constitution and composition, it is further evident that the alumina in the cements high in alumina is combined to form an aluminate in which the ratio of lime to alumina is low. This follows from the fact that these ($CaO.Al_2O_3$ and $3CaO.5Al_2O_3$) react with water

much more slowly than the others high in lime. The other constituents present, $2CaO.SiO_2$ and $2CaO.Al_2O_3.SiO_2$, react very slowly also. It is hardly possible to remove test pieces made of the former from molds within 2 weeks. The latter have not been noted to acquire set at all. In all cases under consideration the setting phenomena noted have been due therefore to the properties of the aluminates or to these properties as modified by plaster. The latter has invariably produced a marked acceleration of both initial and final set. No very satisfactory explanation can be offered in regard to this phenomenon. It has been stated before that the normal hydration of these aluminates consists in the formation of hydrated tri calcium aluminate and hydrated alumina, both separating at early periods in a colloidal form. It is possible that in the presence of the plaster the solubility of the aluminate is increased to a greater degree, and within the same period a greater amount of the colloidal material will separate out, than if it had not been present. This more rapid solution and coagulation would produce a more rapid set.

For determining the strength-developing properties of these aluminous cements, the customary neat and 1:3 standard sand briquettes were made, as well as 2-inch compression cubes of the standard sand mortars and 6 by 12 inch gravel concrete cylinders of several proportions. The majority of these test pieces were made from both the material containing plaster and that not containing the latter. This is particularly true of the small specimens which were made before the concrete specimens, and which showed that the presence of plaster reduced the strength.

The small specimens were generally stored in water after the first 24 hours in the damp closet and the concrete specimens were stored continuously in the latter. However, other work had shown the marked increase in strength if the test pieces were stored in the water for one-quarter of their age and then stored in the air of the laboratory. Hence for certain periods this kind of storage was used in addition to the ordinary storage. This "combined storage" method was used also for the concretes tested at one year, but the storage in the damp closet was somewhat longer, having been six months for the test pieces made from cements 1 to 5, inclusive, and four months for those from 6, 7, and 8.

The data obtained from breaking these test pieces are presented in Tables 6 to 9, inclusive.

TABLE 6.—Tensile Strength of Neat Specimens in Pounds per Square Inch

[C S=Stored for one-fourth of period in water and the remainder in the air of laboratory. W P=Cement contained no plaster. P=Cement contained 3 per cent plaster. B=Broke by handling while placing in the machine]

Cement No.	1	2	3[a]	4	5	6	7	8
Tested at 24 hours: Stored in water—								
W P	455	590		505	320	385	630	345
P	375	315		585	300	385	385	405
Tested at 7 days: Stored in water—								
W P	220	100		715	490	265	955	825
P	790	510		880	475	215	500	290
Tested at 28 days:								
Stored in water—								
W P	130	150		750	505	503	1020	535
P	810	845		955	490	500	475	210
Stored in C S—								
W P	315	385		910	595	280	880	975
P	525	470		815	725	370	680	680
Tested at 90 days: Stored in water—								
W P	200	200		880	545	495	1105	130
P	680	635		725	545	480	905	28
Tested at 26 weeks:								
Stored in water—								
W P	260	120		910	610	530	790	160
P	135	145		820	515	320	470	175
Stored in C S—								
W P	85	50		510	580	360	840	620
P	475	570		500	580	260	885	190
Tested at 1 year: Stored in water—								
W P	185	145		960	665	445	700	130
P	135	165		630	500	415	840	125
Tested at 3 years: Stored in water—								
W P	225	255		1100	640	500	525	B
P	200	185		440	545	375	530	B

[a] No neat test piece made.

TABLE 7.—Tensile Strength of 1:3 Standard Mortars in Pounds per Square Inch (Average of 3 test pieces)

[C S=Stored for one-fourth of period in water and remainder of period in air. P=Plaster added to the cement. W P=No plaster added to the cement]

Cement No.	1	2	3	4	5	6	7	8
Tested at 24 hours: Stored in water—								
W P	370	275	270	415	155	220	345	330
P	380	295	195	355	155	175	250	370
Tested at 7 days: Stored in water—								
W P	403	385	385	280	170	175	295	590
P	360	305	360	335	155	115	225	390
Tested at 28 days:								
Stored in water—								
W P	435	320	525	335	170	280	290	540
P	310	595	345	310	175	100	295	280
Stored in C S—								
W P	730	305	615	345	365	405	560	830
P	720	570	585	415	370	285	515	530
Tested at 90 days: Stored in water—								
W P	440	395	560	375	225	285	350	770
P	465	480	390	355	172	135	370	195
Tested at 26 weeks:								
Stored in water—								
W P	525	390	575	500	300	245	385	590
P	530	375	480	427	163	170	360	380
Stored in C S—								
W P	595	500	900	400	535	605	575	960
P	525	445	665	555	540	315	595	425
Tested at 1 year: Stored in water—								
W P	525	425	610	435	320	295	390	700
P	550	470	490	440	185	140	330	460
Tested at 3 years: Stored in water—								
W P	715	605	560	545	350	335	425	615
P	555	605	335	580	260	155	200	460

TABLE 8.—Compressive Strength of 1:3 Standard Sand Mortars in Pounds per Square Inch (Average of 3 Test Pieces)

[C S = Stored for one-fourth of period in water and the remainder in the air of the laboratory. W P = Cement contained no plaster. P = Cement contained 3 per cent plaster]

Cement No	1	2	3	4	5	6	7	8
Tested at 24 hours: Stored in water—								
W P	3570	4260	2240	4410	1035	1340	2165	2860
P	2860	3050	1830	3380	1060	1060	1870	2710
Tested at 7 days: Stored in water—								
W P	5945	4760	4940	4880	1400	1260	5140	8610
P	2955	4090	2450	2200	1180	940	1720	4290
Tested at 28 days:								
Stored in water—								
W P	3765	3945	6475	5285	1820	1425	3930	7845
P	2790	3535	2540	2965	1160	790	1415	3680
Stored in C S—								
W P	6815	7625	8000	6400	2875	3130	7085	10690
P	4005	5515	4655	5140	2665	2315	3610	7415
Tested at 90 days: Stored in water—								
W P	3945	4735	7180	4565	2070	1680	5205	8260
P	3015	4630	3210	2495	1465	925	2630	4745
Tested at 26 weeks:								
Stored in water—								
W P	5330	4570	7760	3470	2900	1895	3605	6380
P	3860	4850	4630	2270	2200	1475	3080	4605
Stored in C S—								
W P	6015	5290	11050	3480	3845	4345	6090	8045
P	4945	5470	7100	4920	3540	3055	5430	7475
Tested at 1 year: Stored in water—								
W P	3045	4430	5315	3200	2410	2310	5225	8945
P	2850	3580	3015	2355	1720	685	2010	4260
Tested at 3 years: Stored in water—								
W P	3410	4960	5910	3185	3400	2330	3055	8640
P	2815	4110	3115	2730	2785	1345	2480	3285

TABLE 9.—Compressive Strength of 1:1.5:4.5 and 1:3:9 Gravel Concrete in Pounds per Square Inch (Average of 3 Test Pieces)

Cement No.	1[a]	2	3	4	5	6	7	8
Tested at 24 hours: Proportions—								
1 : 6	2825	3000	Soft.	3145	310	840	1475	2930
1 : 12	.....	.....	.....	880	Soft.	275	455	890
1 : 12P	.....	1500	.....	1630	.....	315	375	470
Tested at 7 days: Porportions—								
1 : 6	3270	4150	2175	3570	885	765	4135	6010
1 : 12	.....	.....	.....	1005	225	275	1405	2270
1 : 12P	.....	1590	.....	1435	.....	400	675	1480
Tested at 28 days: Proportions—								
1 : 6	3310	4740	3680	3370	1250	990	4625	7060
1 : 12	.....	.....	.....	1000	365	305	1500	3415
1 : 12P	.....	1890	.....	1415	.....	410	645	1740
Tested at 90 days: Proportions—								
1 : 6	4150	2480	3900	2930	1050	1260	4640	7050
1 : 12	.....	.....	.....	875	390	400	1600	3980
1 : 12P	.....	1580	.....	1280	.....	700	800	1760
Tested at 26 weeks: Proportions—								
1 : 6	4180	1760	4525	2565	1270	1515	4450	7610
1 : 12	.....	.....	.....	800	550	470	1580	3910
1 : 12P	.....	1580	.....	1385	.....	570	600	1700
Tested at 1 year: Proportions—								
1 : 6	3310	1760	4135	2565	1460	1440	4650	6720
1 : 6 [b]	5540	2360	5660	3280	2110	2035	5300	8220
1 : 12	.....	.....	.....	860	570	470	1350	3930
1 : 12 [b]	.....	.....	.....	1275	870	810	1815	4445
1 : 12P	.....	1530	.....	1525	.....	600	655	1470
1 : 12P [b]	.....	2250	.....	2480	.....	1055	1080	2450
Tested at 3 years: Proportions—								
1 : 6	2080	1835	4720	2415	2035	1700	3400	5675
1 : 6 [b]	2230	2160	5060	3050	2005	2215	3840	7540
1 : 12	.....	.....	.....	900	875	1055	1780	2670
1 : 12 [b]	.....	.....	.....	1180	910	865	1865	3915
1 : 12P	.....	1605	.....	1860	.....	865	855	1400
1 : 12P [b]	.....	1695	.....	2160	.....	1145	1080	2470

[a] Cement used in making specimens for cement 1 contained 3 per cent plaster, also all specimens in lines headed 1 : 12P were made from cement containing 3 per cent plaster. None of the other cements contained plaster.

[b] These specimens were stored first in the damp closet (for the first 6 months in the case of cements 1–5, inclusive, and 4 months for cements 6–8) and for the remainder of the period in the air of the laboratory. All other specimens were stored in the damp closet for the entire period aged.

As usual it will be seen from these that the cements do not give a neat tensile strength which seems to bear any relation to the strength which the same cement will develop as a mortar or a concrete. Such is the case generally also with Portland cement, and consequently the Government revised specifications [10] for the latter material do not recognize nor require a neat test piece. The following data (Table 10), selected from Tables 6 to 9 and showing the strength at the end of one year for certain test pieces made from the cements containing no plaster, will illustrate this.

TABLE 10.—Strengths (at the End of One Year) of Cement Test Pieces Containing no Plaster

Specimen and test	Strengths for cement No.—				
	2	4	6	7	8
	Lbs./in.²	Lbs./in.²	Lbs./in.²	Lbs./in.²	Lbs./in.²
Neat, tension	145	960	445	700	130
1:3 mortar, tension	425	435	295	390	700
1:3 mortar, compression	4430	3200	2410	5225	8945
1:6 concrete	1760	2965	1440	4650	6720

At the same time it can not be stated that Table 10 shows any relation between the strengths developed by the same cement in any of the four forms of test pieces used. That there should not be any relation between the tensile strength and the compressive strength is not surprising when the marked difference in the shapes of the tension and compression test pieces is considered. The former have been analyzed and discussed to a considerable degree,[11] and it has been shown that such a shaped test piece does not satisfactorily distribute the stresses applied to it during the breaking in a testing machine. This is especially true of such a rigid material as set cement, or bodies, such as mortars, containing it.

There must also be borne in mind the actual difference in the materials of which the test pieces are made. The differences in physical properties between a neat cement test piece and one made of mortar are generally conceded. There is also reason to grant a difference between a mortar and a concrete specimen. The methods of making the former are such as would produce a body differing from the latter. Thus the constituents of the mortar

[10] U. S. Government Specification for Portland Cement, B. S. Circular No. 33.

[11] J. B Johnson, Materials of Construction, edition 4, page 435. Coker, The distribution of stresses at the minimum section of a cement briquette, paper 284, section 2, Proc. Internat. Soc. for Test. Mat.; 1912.

are kneaded and rubbed together in the mixing and usually they are in actual contact with the hand; the concrete, however, is only mixed together by various implements in the hands of the operators. In molding there is again a marked difference of procedure, the mortar being placed in the mold in relatively much smaller quantities and subjected to much more compacting. Neglecting the character of the large aggregates and the difference in the degree of hydration which takes place in the several types of specimens, it would seem that a mortar should differ both from a concrete and from a neat cement also on account of the difference in cohesive and adhesive properties developed by the different cements in setting. Thus the formation of much colloidal material alone would develop the property of the cement to bond foreign materials (fine or coarse aggregate) together (adhesive strength) as opposed to the marked formation of crystalline material replacing some colloidal material. This latter would bond together the hydrated cement itself, as in the neat specimens (cohesive strength), but would not add much to the strength of the hydrated cement as a bond of the aggregate. Thus a hydrated neat specimen containing relatively much colloid would have a comparatively low strength when compared with one containing more crystalline products of hydration. This different character of hydration would, as measured by the strength of the richer sand specimens, possibly be but slight, while in a concrete the cement with a low neat strength would give a high strength. Consequently it would not seem as if there should be any fixed relation between the strength developed by cements as neat, mortar, or concrete test pieces. This does not always appear to be true in respect to Portland cements. But it should be remembered that the majority of the latter do not differ much among themselves in composition or physical properties, and consequently those which develop high testing mortars also develop high testing concretes at the same periods. When, however, extremes in compositions or physical properties are compared, then it is found that one producing a high-strength mortar may produce a low-strength concrete, and vice versa.

A noticeable feature is brought out when the tensile specimens of neat and mortar are compared. It is readily seen that the mortars of the cements high in alumina (Nos. 1, 2, and 8) at comparatively early periods develop higher strengths than the neat specimen. This becomes more marked at the late periods. At the three-year periods, cements Nos. 1, 2, and 8 without plaster

develop in this order, as neat test pieces, tensile strength of 225, 255, and 000 pounds per square inch. As 1:3 mortar at the same period they develop tensile strength of 715, 605, and 615 pounds. An examination of Table 6 reveals that the neat test pieces of these three cements developed their highest strength at either 24 hours or 7 days, after which there was a retrogression, whereas the mortars have gained strength rather consistently. It would appear as if the hydration of the former had been very complete at those early periods. Their continued storage in water had permitted of the absorption of water by the hydrated colloidal material to such a degree that the strength was decidedly reduced. Such an action would be similar to the absorption of excess water by fish glue or any other organic glue whereby the cementing properties of these are reduced to a minimum or destroyed completely. On the other hand, the hydration of the mortars was less, access of water to the interior of the test piece having been slower on account of the impervious nature of the sand grains.

The other cements do not show this unusual condition of a mortar having a greater strength than the neat cement from which it was made. However, these others contain silica, combined either as the orthosilicate or as the ternary silica-lime-alumina compound, $2CaO.Al_2O_3.SiO_2$. The former hydrates very slowly and to but a comparatively slight degree, although at the end of a year it will have attained almost as great a strength as a normal Portland cement. The ternary compound has not been noticed to harden in this laboratory. The gain in strength of the neat specimens of these cements containing the larger amounts of silica is due therefore to the hydration of the orthosilicate, which manifests itself at the late ages and offsets the loss in strength of those containing but the small amounts of silica.

When discussing the matter of "setting," there was pointed out the marked effect produced on the aluminates by the use of plaster of Paris. The latter caused such a decided acceleration of the time of set as to render the cements almost unusable. The use of this material also produced the undesirable effect of reducing the strength. This is not very marked with certain forms of test pieces, as the tension briquettes, or with certain cements, as those high in silica. Also the effect was not as marked at some periods as others, as shown by the mortars tested at early periods in compression. But generally the plaster was undesirable not only on

account of producing too quick a set but also usually reducing the strength.

The principal feature of the strength data, however, is the high strength developed at early ages by the mortars and concretes, especially by those cements high in alumina. Those containing the larger amounts of silica do not show such high initial strengths but, on the other hand, show a greater increase in strength with age. The ideal composition considered both from economy in cost of raw materials and general qualities of the cement produced therefrom would appear to be exhibited by cements No. 4 or 7. Their composition is such as would permit of the use of relatively impure alumina or limestone, or the use of a quantity of kaolin or low iron-oxide clay in place of all alumina. The resulting product would have the desired early hardening property in addition to the one of exhibiting a continuous increase in strength over a long period.

The cause of the high early strength and the relatively small increases with age can be inferred from previous remarks dealing with the hydration of the aluminates. This hydration is not only rapid but more thorough than with Portland cements. There is therefore produced more of the cementing agent, which in this case is both the hydrated tricalcium aluminate and the hydrated alumina, both in a colloidal form and comparatively insoluble. In the case of Portland cement the only aluminate present—the tricalcium aluminate—does not split off any of the hydrated alumina, and the silicates form a hydrated silicate and lime hydrate. The latter, compared with hydrated alumina, is relatively soluble, and when it does separate out of solution it forms the crystals so noticeable in specimens of set Portland cement. Therefore in the latter cements only part of the hydrated compounds formed have cementing properties, while with the aluminates, all of the products of hydration not only have cementing properties but they are furthermore developed very rapidly. This difference not only accounts for the difference in the early strength but also for the rate of increase in strength.

A microscopical examination was made of the various specimens after breaking at the three-year period. In general, it may be said that the largest amounts of unhydrated cement were noticed in the neat specimens, the next in the sand specimens, and the least amount in the concrete. The concrete stored in the air was markedly different from that stored in the damp closet,

the difference being plainly evident macroscopically in the more chalky appearance of the former. Also under the microscope the former showed more unhydrated cement, and the hydrated products were more amorphous or else the crystals were much more poorly developed. The sand specimens also showed a better crystallization than the neat ones, but possibly not as much as the concrete cured in the damp closet. The crystalline products were the tricalcium aluminate, a little hydrated lime in some cases, and some tricalcium sulphoaluminate. At the end of three years the ternary compound ($2CaOSiO_2Al_2O_3$) was still unhydrated, and in several cases the orthosilicate was noticed only partially hydrated. The same was true of the monocalcium aluminate. The amount of tricalcium sulphoaluminate was strikingly small even if the cement had been mixed with plaster. It was noticed usually in cavities and in specimens made from cement No. 5 in particular. It was also noticed in several specimens made from unplastered cement in cavities near the exterior where undoubtedly the sulphate had been taken from the tap water in which they were stored. The crystals were long but very small in cross section and appeared in tuft-like masses which had no mechanical strength. Many times the amount actually noted in any cavity would have been necessary before any disruptive force could have been generated.

The fact that the concretes contained more crystalline products of hydration than the neat specimens should not be used as an argument against the theory advanced above to explain the difference in the adhesive and cohesive properties developed by a cement. While the concrete did contain more crystalline hydrated material, the cement as a whole was also more completely hydrated, so that the percentage of crystals in the hydrated material was actually less in the concretes than in the neat test pieces.

SUMMARY

There have been produced in a rotary kiln certain cements, characterized by a high alumina content, which have the property of hardening in a short time to such a degree as to give very high strengths. This is especially true when the cements are used in mortars or concrete.

The temperatures necessary for burning the raw mixes and the actions of these in the kiln were little if at all different from those observed in the burning of Portland cement.

The raw materials used were limestone and calcined alumina; in some cases the effects of such impurities as silica and iron oxide were studied by the use of kaolin, bauxite, or flint, in addition to the above.

The ground clinker in some cases set very rapidly, but in other cases where the alumina was very high it set slowly.

The hardening was very rapid in all cases unless the content of impurities became very high, when the hardening resembled more closely that of Portland cement.

The predominant constituent noted in the clinker was the monocalcium aluminate or the 3:5 calcium aluminate. As the silica increased, increasing amounts of the orthosilicate of lime or the ternary compound $2CaO.SiO_2.Al_2O_3$ were noted.

The products of hydration were hydrated tricalcium aluminate and hydrated alumina resulting from the aluminates, and hydrated lime from the orthosilicate of lime, leaving the latter in a hydrated form but of a lower lime content than that required to form $2CaO.SiO_2$.

The early hardenings were due to the rapidity with which the aluminates hydrate. The high strengths were due to the thoroughness of the hydration and to the fact that this was largely colloidal.

The methods of storage of the test pieces showed that these cements had their strengths rather materially reduced by the absorption of moisture. This was due to the large amount of colloidal products of hydration, which are very susceptible to moisture changes. They would therefore not be satisfactory for use when subjected to the action of water.

The cost of the raw materials at the present time is such as to preclude the commercial use of these cements except under very special cases. They are of much interest, however, in showing that besides the high-limed silica compounds there is another group of compounds in the lime-alumina-silica system which has cementing qualities. This investigation showed, however, that these two groups are the only ones of this character in the system.

WASHINGTON, February 14, 1921.

DEPARTMENT OF COMMERCE

TECHNOLOGIC PAPERS

OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 198

RESULTS OF SOME TESTS OF MANILA ROPE

BY

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SEPTEMBER 15, 1921

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RESULTS OF SOME TESTS OF MANILA ROPE

By Ambrose H. Stang and Lory R. Strickenberg

ABSTRACT

This paper summarizes the results of tensile tests of 368 specimens of manila rope. Most of the material was submitted on purchase orders for Government departments. They were all 3-strand, regular lay manila rope having diameters from ½ inch to 4½ inches.

A summary of the results is given in tables and graphically. A formula is given for determining the average breaking load as a function of the diameter of the rope. The test results cover sufficient range and show such consistency that it is believed that the formulas may be used safely for 3-strand, regular lay manila rope of the sizes indicated. The ropes showed a continually varying modulus of elasticity and no well-defined proportional limit.

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I. INTRODUCTION

The results of tests on manila rope discussed in this paper represent some of the data obtained at the Bureau of Standards during the past few years. The tests were made under the supervision of J. H. Griffith, J. G. Bragg, W. H. Virgin, and others.

Most of the specimens were submitted by various rope manufacturers on purchase orders for Government departments.

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Although the purchase specifications until recently were not identical for all departments, a fixed procedure was adopted by the laboratory for all rope tests. On April 4, 1918, a standard specification for manila rope was adopted at a joint conference between the representatives of the various Government departments and of the rope manufacturers. This has since been used by many of the departments, as, for example, by the Panama Canal (No. 307–C, Standard Specifications for Manila Rope, 1920).

As the method of testing adopted by the laboratory was the same as that required by the standard specifications (paragraph 9), these results may be considered representative of those which would be obtained on commercial rope if tested under the Government standard specifications. Due to the fact that the tests were not part of an investigational program, the effect of many of the variables on the properties of rope was not determined.

II. SCOPE OF TESTS

The samples of rope ranged in diameter from ½ to 4½ inches, inclusive, and consisted of commercial 3-strand, regular lay ropes.

The breaking loads were observed in all the tensile tests, and observations were made upon a considerable number of the samples for weight per linear foot, number of yarns, and the "lay" of the rope and strands. Measurements of elongation were also made upon 18 specimens of special "hard-laid" ropes, ¾, 1¼, 1½, and 1¾ inches in diameter.

III. CONSTRUCTION OF ROPE

1. NUMBER OF YARNS

All kinds of rope and cordage are composed of 1 or more yarns, which are also known as plys or threads, depending upon the kind of rope in which they are used. In regular lay 3-strand rope the number of these yarns may be expected to vary with the diameter; that is, a rope of large diameter will have more yarns than one of small diameter if the yarns are the same or about the same size in each rope. This investigation showed that ropes of small diameter, as a rule, have smaller sized yarns than those of large diameter and that even those of the same diameter from different makers vary somewhat in the number. The yarns were counted in many of the ropes, the results being given later in the paper.

FIG. 1.—*One lot of manila rope samples representing a year's purchase of rope for one storage depot of the U. S. Army*

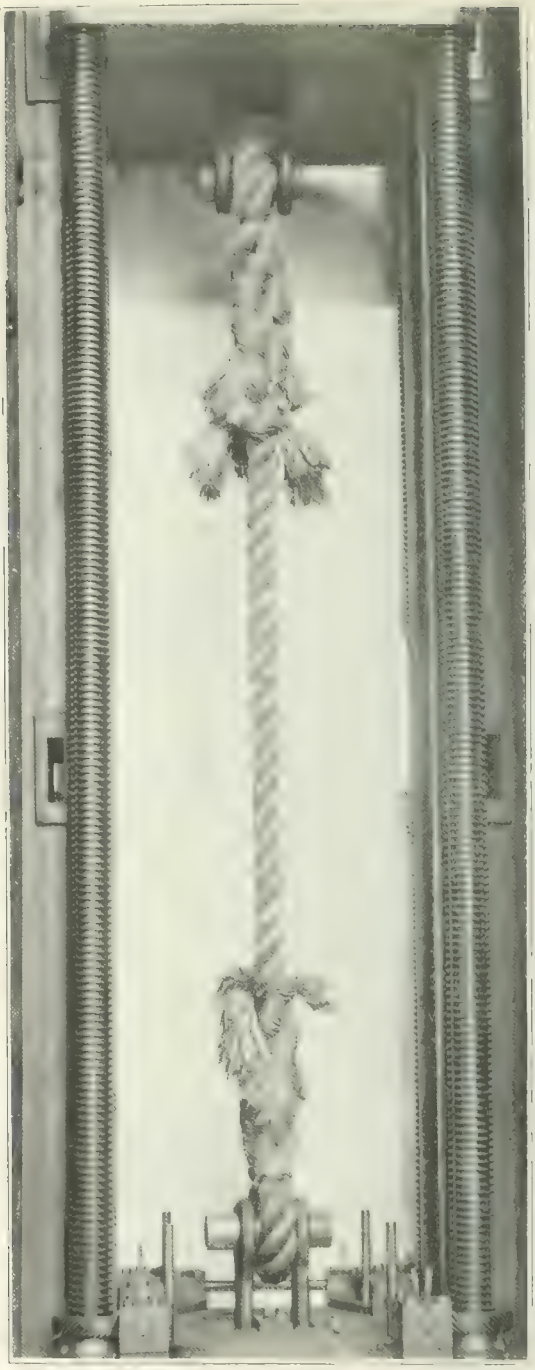

FIG. 2.—*A rope specimen in position for tension test in a 600 000 pound testing machine*

2. LAY OF ROPE AND STRANDS

The distance along the axis of the rope in which the rope makes one complete turn is called the "lay" or "jaw" of the rope. The distance along the axis of the strand in which the yarn makes one complete turn is called the "lay" of the strand. When the yarns twist in one direction about the axis of the strands and the strands twist in the opposite direction about the axis of the rope, the rope is known as regular "lay." The hardness of a rope is determined by the amount of twist given to the strands, and consequently it may be referred to as hard or soft laid.

IV. METHOD OF TESTING

1. PREPARATION OF ROPE

All specimens for tensile tests were prepared with an eye splice in each end, the inside diameter of the eye being about 7 inches, and the length between splices not less than 5 feet. Splices were carefully made in an effort to cause the specimen to break in the body of the rope. To prevent slipping at the splice, the specimens were soaked in water overnight. However, in spite of the precautions, about 50 per cent of those tested broke at or near a splice.

2. MACHINES USED FOR TESTING

Most of the specimens were broken in a 600 000 pound testing machine, although a few of the smaller sizes (under 1 inch in diameter), for which the distance between the splices would permit, were broken in a 100 000 pound machine. The machine was operated so that the speed of the moving head was not more than 4 inches per minute. Fig. 2 shows a rope specimen in position in the larger machine.

3. DESCRIPTION OF TESTS

(*a*) TENSION.—Fig. 2 shows the method of fastening the ends of the rope in the machine. The beam of the weighing apparatus was kept balanced as the load was gradually applied, and the breaking load was indicated on the beam at the completion of the test.

(*b*) WEIGHT.—To determine the weight of the rope, the specimen was subjected to a load of (P) pounds, equal to 200 times the square of the diameter of the rope in inches ($P = 200\ D^2$). With the specimen under this load, a certain length was marked

off on the rope, and this length was then cut out of the specimen after the load was removed. This portion was weighed, and the weight per foot was calculated.

(*c*) SIZE.—The size was determined on the rope specimens when under the same load as was used in determining the weight. The circumference was measured by passing a fiber snugly around the rope and cutting it at the point of intersection. The length of fiber was then measured to obtain the circumference. The lay, in number of turns per foot, of both rope and strands was also measured directly upon the rope specimen, and the number of strands and yarns was counted.

V. RESULTS OF TESTS

TABLE 1.—Tests of Manila Rope. (Sample Log Sheet)

Sample number	Length		Diameter	Weight of sample		Weight of rope	Number of strands
	Feet	Inches	Inches	Pounds	Ounces	Lb./ft.	
1	15	0	1 1/4	4	1.5	0.273	3
2	14	10	1 1/8	3	9.2	.241	3
3	15	9 1/2	1 1/8	4	[illegible]	.265	3
4	14	6	1 1/4	3	15.2	.272	3
5	14	9	1 1/8	3	15	.267	3
6	14	10	1 1/4	3	15.8	.269	3
7	15	1 1/2	1 1/4	4	8	.298	3
8	13	7	1 1/8	3	8	.258	3
9	14	[illegible]	1 1/8	4	8.5	.309	3
10	15	0	1 1/4	4	5.8	.291	3
11	15	1 1/2	1 1/8	3	15.5	.263	3
12	14	8	1 1/8	3	11.2	.252	3
13	14	8	1 1/8	3	10	.247	3

Sample number	Number of yarns in rope	Lay of rope	Lay of strand	Lay of yarn	Breaking load	Remarks [a]
		Turns/ft.	Turns/ft.	Turns/ft.	Pounds	
1*	54	3.7	3	7.2	13 600	
2	74	3.7	3.8	8.7	9 950	Marked "[illegible] pure manila"
3	69	4	3.5	9.2	10 500	
4	96	3.3	3	11.3	11 700	Two strands broke near splice
5	54	4	4	7.2	9 500	
6	99	3.7	3.5	10.2	10 500	Two strands broke about 6 inches from splice
7	85	3.7	3	9.9	10 800	
8	83	4	3.2	7.9	11 700	
9	72	4	3.2	9.3	8 700	
10	84	3.7	3	9.8	10 500	
11	84	[illegible]	3.5	7.8	10 700	
12	87	3.5	4	10.4	9 950	
13	90	3.3	3.8	8.7	10 400	

[a] Except where noted, one strand of each rope broke at or near splice.

TABLE 2.—Breaking Loads for Manila Rope

Dimensions of ropes in inches		Number of tests	Observed breaking loads			Values of breaking load calculated from $L = cd(d+1)$ $c = 6300 \ldots\ldots 3700 \ldots\ldots 5000$			Government standard specifications
Diameters	Circumferences		Maximum	Minimum	Average	Maximum	Minimum	Average	
			Pounds	Pounds	Pounds	Pounds	Pounds	Pounds	Pounds
½	1 57	23	3 920	2 300	3 250	4 725	2 775	3 750	2 450
⅝	[illegible]	17	6 550	4 250	5 300	6 400	3 750	5 100	4 000
¾	2 2	14	7 020	5 180	6 350	8 270	4 850	6 565	4 900
13/16	2 90	26	8 730	5 850	7 100	9 275	5 450	7 360	5 900
1	3 14	72	12 700	7 800	9 770	12 600	7 400	10 000	8 200
1 ⅛	3 5	[illegible]	12 900	8 700	11 480	15 060	8 850	11 950	11 000
1 ¼	3 93	76	17 600	10 420	13 650	17 725	10 410	14 060	12 500
1 ⅜	4 71	48	24 800	13 280	18 440	23 630	13 875	18 750	17 500
1 ½	5 07	14	25 210	16 410	21 830	26 875	15 800	21 325	21 500
1 ¾	5 5	14	31 130	20 000	24 400	30 325	17 800	24 060	25 500
2	6 28	5	27 200	23 140	26 140	37 800	22 200	30 000	30 000
2 ¼	7 85	[illegible]	54 700	32 300	44 000	55 100	32 350	43 750	43 500
3	9 42	[illegible]	74 000	45 000	59 300	75 600	44 400	60 000	61 000
3 ½	10 10	17	105 000	58 100	85 800	99 230	58 275	78 750	.....
4	12 57	2	115 700	87 500	101 600	126 000	74 000	100 000	.....
4 ½	14 14	1	110 000	110 000	110 000	155 900	91 600	123 750	.....

TABLE 3.—Number of Yarns in Manila Rope

Diameter of ropes in inches	Number of tests	Number of yarns in rope			Number calculated from formula $N = 50d(d+0.4)$
		Maximum	Minimum	Average	
½	21	21	18	20	22
⅝	15	36	30	33	32
¾	11	39	39	39	43
13/16	26	51	39	42	49
1	70	96	54	72	70
1 ¼	76	162	72	105	103
1 ½	46	225	129	144	143
1 ⅝	12	225	150	174	165
1 ¾	12	201	150	189	188
2	5	234	150	222	240
2 ½	8	354	315	339	362
3	22	582	435	504	510
3 ½	16	720	645	690	683
4	2	915	846	885	880

VI. DISCUSSION OF RESULTS

The average tensile strengths of 368 specimens of manila rope, ranging from ½ to 4½ inches in diameter, are shown in Table 2, and have been plotted in Fig. 3 as a function of the diameters of the ropes. The breaking load is approximately a quadratic function of the diameter, and may be closely expressed by the formula:

$$L = cd(d + 1),$$

where L is the breaking load in pounds, d the diameter of the rope in inches, and c is a constant that is equal approximately to 6300

for the strongest ropes tested, 5000 for the average, and 3700 for the weakest ropes. In other words, with the observed loads from the individual tests plotted for the various diameters of ropes as

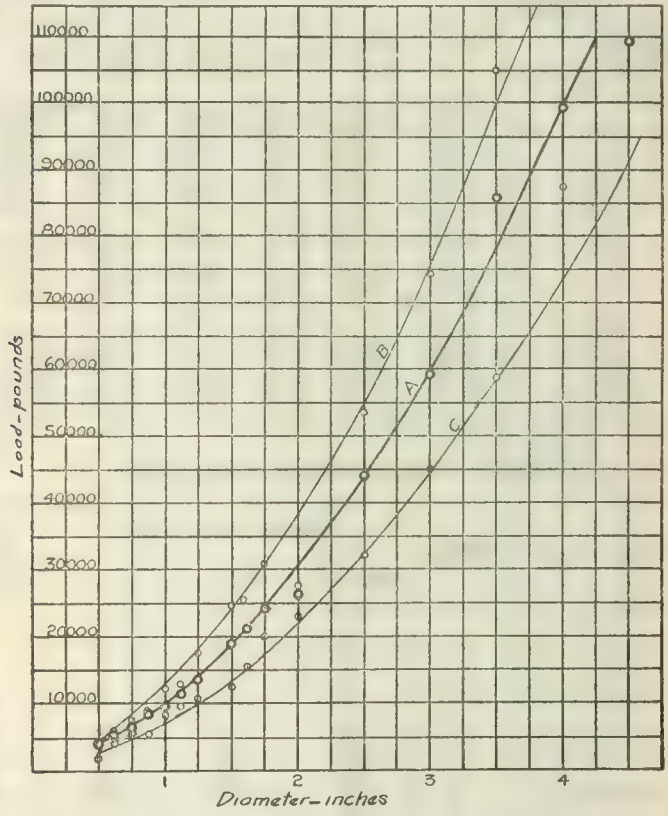

FIG. 3.—*Relation between breaking load and diameter of manila rope*

The plotted points show the close agreement of the experimental values with those computed from the formulas

shown in Fig. 3, the curve for the average results conforms very closely to the equation (see curve *A*, Fig. 3):

$$L = 5000\ d\ (d+1).$$

The minimum results are very closely expressed by the equation (see curve *C*, Fig. 3):

$$L = 3700\ d\ (d+1);$$

and the maximum values are expressed by the equation (see curve *B*, Fig. 3):

$$L = 6300\ d\ (d+1).$$

These equations apply only to 3-strand, regular lay manila rope over ½ inch in diameter.

It should be noted (see Table 2) that the average values given by the formula:

$$L = 5000\ d\ (d+1),$$

and from the tests agree closely with those obtained from the standard specifications. An attempt to plot all three of these values in

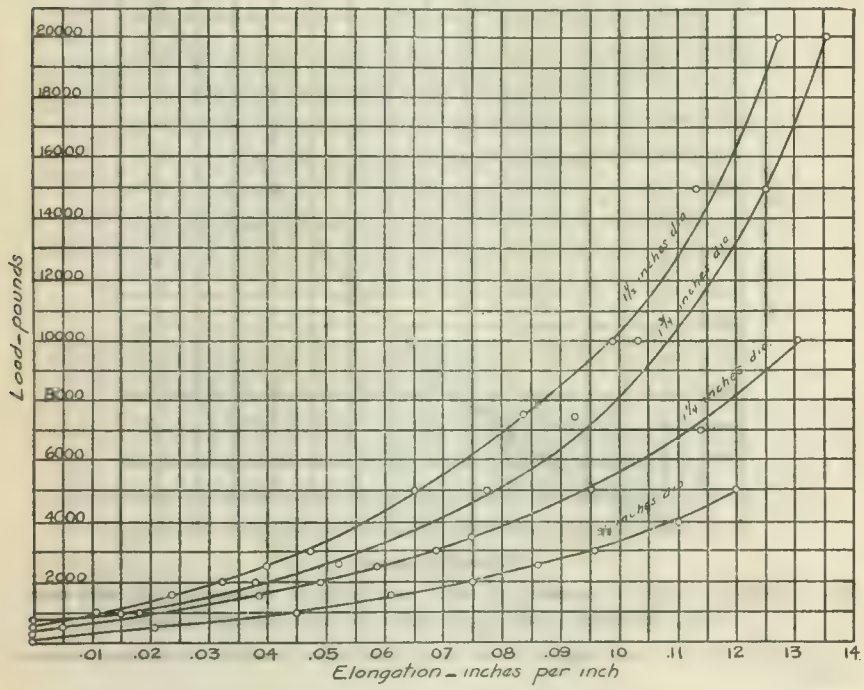

FIG. 4.—*Load-elongation curves for hard laid ropes of various diameters*

The plotted points are the averages of the observed elongations for specimens having the same diameter, the highest value in each case being the average breaking load

Fig. 3 showed that the curves practically coincided with the curve *A* which was drawn.

In Fig. 4 load-elongation curves have been plotted for 18 specimens of hard-laid rope of 4 different diameters. The plotted points are the averages of the observed elongations at a given load for specimens having the same diameter, the highest value in each case being the average breaking load. It is readily noted from the curves that rope has a variable modulus of elasticity and no well-defined proportional limit.

Table 3 shows the relation between the number of yarns in a rope and the diameter of the rope. The number of yarns was counted, and the maximum, minimum, and average number found in the different sized ropes is given here. These values have been plotted in Fig. 5 against the diameter of the rope. The average

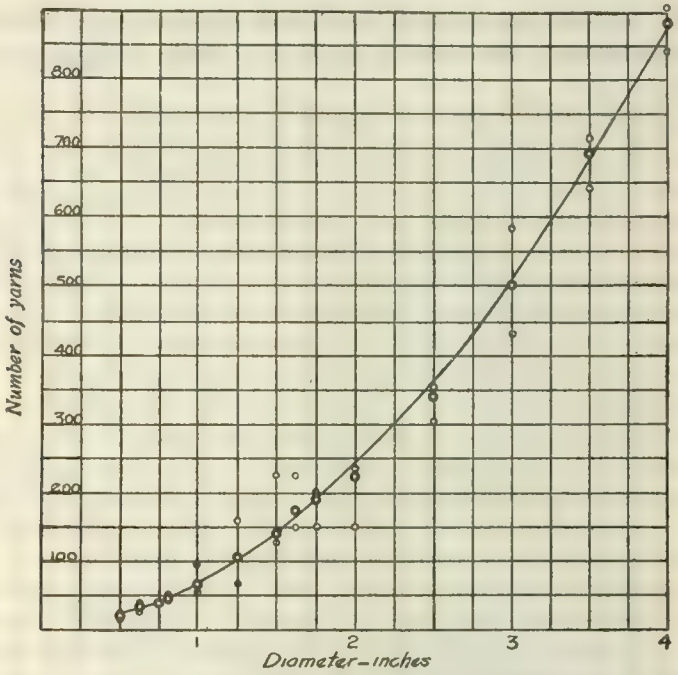

FIG. 5.—*Relation between number of yarns and diameter of manila rope with maximum, minimum, and average values for each diameter shown*

Large circles indicate average observed values; small circles indicate maximum or minimum observed values

number of yarns composing a rope may be approximately represented by the formula:

$$N = kd\ (d + 0.4),$$

where N is the number of yarns, k is a constant equal to 50, and d is the diameter of the rope in inches. The last column in Table 3 shows the number of yarns as represented by this formula. This formula is represented also graphically by the curve of Fig. 5, from which the agreement between the experimental values and those obtained from the equation may be judged.

As stated, this is only approximate, since the actual number of yarns composing a rope may vary as much as 30 per cent

either above or below the number expressed by this empirical equation. However, it gave results in this work having a probable error of about 10 per cent for the average values.

VII. SUMMARY

1. The average breaking load was found to be approximately a quadratic function of the diameter of the rope. It is expressed quite closely by the equation:

$$L = cd\ (d + 1),$$

in which L is the load in pounds, c is a constant equal to 5000, and d is the diameter of the rope in inches.

2. The ropes showed a continually varying modulus of elasticity and no well-defined proportional limit.

3. The number of yarns composing a rope may be expressed approximately by the equation:

$$N = kd\ (d + 0.4),$$

where N is the number of yarns, k is a constant equal to 50, and d is the diameter of the rope in inches.

4. The test results cover sufficient range and show such consistency that it is believed that the formulas deduced may be used safely for 3-strand, regular lay manila rope for sizes of rope between ½ and 4½ inches in diameter.

WASHINGTON, February 26, 1921.

DEPARTMENT OF COMMERCE

TECHNOLOGIC PAPERS

OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 199

METHOD FOR PRECISION TEST OF LARGE CAPACITY SCALES

BY

C. A. BRIGGS, Associate Physicist

E. D. GORDON, Associate Physicist

Bureau of Standards

SEPTEMBER 24, 1921

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1921

METHOD FOR PRECISION TEST OF LARGE CAPACITY SCALES

By C. A. Briggs and E. D. Gordon

ABSTRACT

This paper outlines a scientific and systematic method used by the Bureau of Standards for testing railroad master and grain hopper scales. A pointer and scale are arranged for reading the position of the beam, and the errors of the scale are determined from observations made upon the freely swinging beam. The procedure of the test is explained with the aid of a record form and computation sheet which was developed in connection with the successful application of the method in the field. In the interest of a uniform and efficient method the scheme outlined is recommended for adoption by those who have occasion to carry out tests on large scales where accuracy of a high order is required.

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1. INTRODUCTION

In making tests of unusual accuracy on compound lever scales, such as grain hopper and master scales, it is the practice of the Bureau of Standards to employ a method for determining the position of equilibrium of the beam similar to that generally used in laboratories on equal-arm precision balances. This method was introduced by one of the authors when the work of standardizing large weights and master scales was first undertaken, and as the

result of several years' application in the field it has now reached an improved routine form. The chief feature of this method is that observations are made by means of a pointer attached to the trig loop, readings being made on a ruled scale mounted on the freely swinging beam.

The application of this method in the field has aroused the interest of the practical scale man engaged in regular scale testing, and practically all who have seen tests carried out in this manner have expressed their desire to become familiar with it, so that they can use it for their own work when the occasion arises. It is in response to this demand for a detailed explanation of the scheme that this description has been prepared and issued.

2. NOT SUGGESTED FOR GENERAL USE IN ORDINARY TESTING

This method is not suggested for use in the ordinary testing of railroad track or motor truck scales or for routine work of this character. It has yet to be determined, however, to what extent this plan can be used to advantage in the regular testing of grain hopper scales. At first it was thought that it should not be used for the latter, but more recent considerations have suggested that it may be desirable to use the method to some extent in this work.

3. ADVANTAGES OF THE METHOD.

The advantages of the method are several: First, it permits tests of scales to be conducted in a systematic and scientific manner. A complete record is made of all observations, so that nothing is left to uncertain or delayed entries. Second, the personal equation and unconscious prejudices of the observer are reduced to a minimum. Third, results may be checked as the observations are made, so that omissions become very apparent and mistakes can generally be discovered and often corrected with reasonable reliability. Fourth, the method in practiced hands enables the test to be carried out much more rapidly than is the case where the ordinary plan is used. It is not necessary to add or remove small weights until by repeated trials the beam comes to a position of equilibrium in the middle of the loop, but the weights required to bring this balance about can be determined from computations derived from observations on the swing of the beam for only two different applications of small weights. The saving of time is particularly important in scales requiring the greatest accuracy, as in these scales changes which take place during the test, on account of the effects of temperature varia-

tions and from a slow settling of the parts, may cause an appreciable part of the total error. Fifth, further advantage lies in the fact that when disturbing conditions are present, as air currents and drafts, it is possible to obtain data which otherwise could not be secured—the readings of the pointer can be watched and recorded and quiet intervals can be recognized and used.

4. DISADVANTAGES OF THE METHOD

The disadvantages are that computations have to be made before the results are available in definite figures, and under the conditions of the test the very fact that time is saved and the work permitted to proceed rapidly may develop a tendency to delay working out the data. This is particularly true in the case of those who have used the method for a short time only and who have not the practice necessary to make the required computations quickly while the test is being made. A slide rule is an almost indispensable item of the equipment, and ability to use it with accuracy and speed is necessary. The very thoroughness with which the test can be carried out also entails a large amount of clerical work.

5. REST POINT DEFINED

The term "rest point" will be defined as the position in which the beam would come to rest if the effects of friction were eliminated. It is the position about which the beam oscillates as a center when vibrating freely.

6. MEANS FOR MEASURING THE POSITION OF THE BEAM

Consider a ruled scale mounted vertically on the beam near the end in a manner such that the position of the beam in its play in the loop can be read by means of a pointer attached to the trig loop or stand. This is shown in Fig. 1. Any position of the beam—as, for instance, the position of equilibrium or rest point—can now be expressed in terms of the reading of the pointer on the ruled scale. The method outlined in this paper for making the observations is based on this fact.

7. SENSIBILITY RECIPROCAL

Likewise, the sensibility reciprocal of the beam, or *sr*, as it will be written, can be expressed as the change in the load required on the platform to change the position of equilibrium of the beam one division as read on the graduated scale. In the particular

method outlined in this paper for making the tests with a maximum economy of time the corresponding value needed for the computations is the weight required to change the position of equilibrium of the beam one-half a division, or one-half the *sr*. This is a consequence of the scheme employed for making the computations in which the rest point is taken as the sum of the readings of the two extreme positions of the beam as it swings freely. This is to render it unnecessary to divide by 2, thereby simplifying the computations, as will be more fully discussed later.

If the weighing mechanism is working properly, the *sr* will be practically constant for any particular load for all positions of the beam.

8. COMPUTING WEIGHT REQUIRED FOR PERFECT BALANCE

Consider the pointer and ruled scale so arranged that when the beam is in a horizontal position in the center of the loop, as for a perfect balance, the reading is 20. Suppose, further, that for a certain load the position of equilibrium of the beam is given by reading 16, and that the ruled scale is arranged to give increasing readings as the beam changes from a low position to a high position. Let also the *sr* be 2 pounds per division. In this case the pointer is 4 divisions below the center of the loop or the mid-position of its play; that is, 16 is 4 divisions below 20. In this case it is easy to see that 4×2, or 8 pounds additional weight, would be required on the platform to bring the beam to a perfect balance, where it would read 20.

From this it can be understood that when the *sr* is known and the position of equilibrium is determined in reference to the ruled scale it is possible to compute the change in weight required on the platform to perfect the balance, and it is unnecessary to take the time and trouble actually to change the weight on the platform.

9. DERIVATION OF DATA FROM MOVING BEAM

In what has just been outlined reference has been made to the reading of the pointer which corresponds to the position of equilibrium or rest point, as it is termed. An apparent way of obtaining such a reading would be to allow the beam to come to rest and then to make the reading. This, however, would not give practical results, as in many cases the position assumed by the pointer would be affected by friction. If friction were not appreciable, the beam would be a very long time in coming to rest.

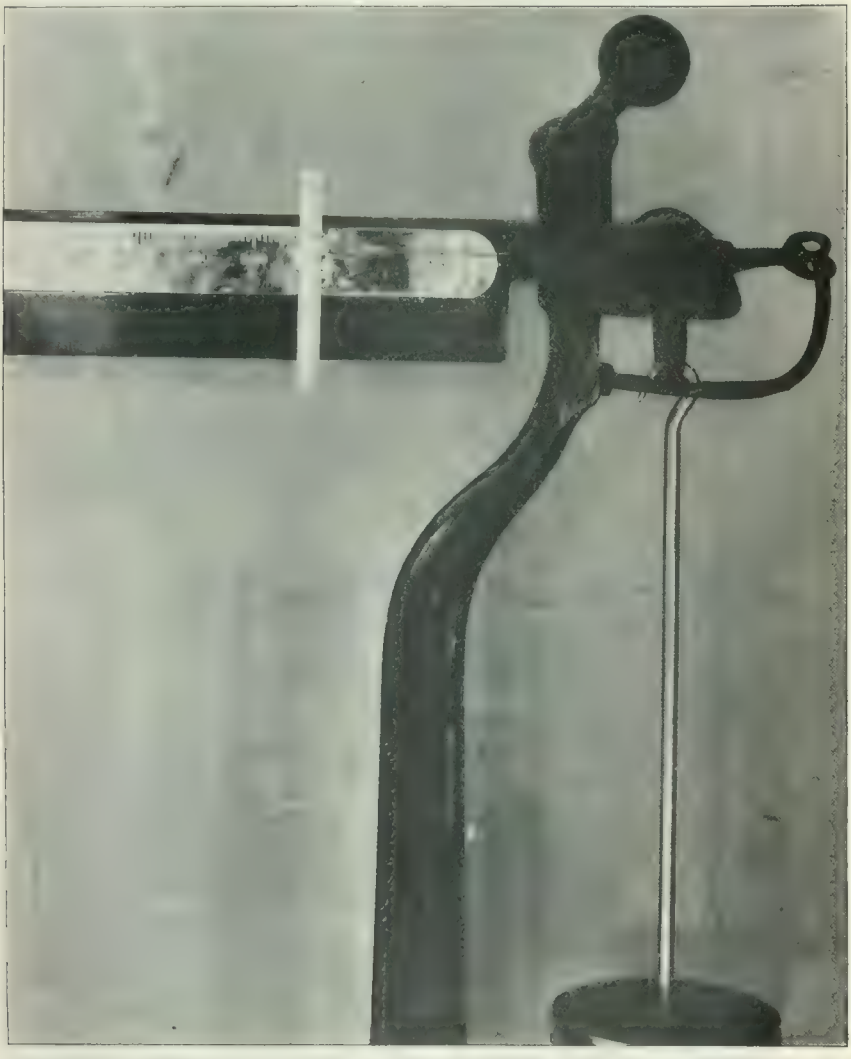

FIG. 1 *View showing pointer attached to trip loop reading on ruled scale mounted on the beam, ready for conducting precision test of a scale*

Suppose that the movement of the pointer over the scale is observed and that there is no friction present. The beam will then swing up and down indefinitely, and the position of equilibrium will be midway between the upper and lower readings of the pointer on the ruled scale. Under these conditions the value can be obtained by taking the upper and lower extreme readings, adding them, and dividing by 2.

10. METHOD FOR ELIMINATING THE EFFECTS OF FRICTION

The friction which affects the movement of the beam during its swing can be eliminated by a simple method. The average of the readings from two successive swings in one direction is taken and added to the reading of the swing occurring between in the other direction. This result divided by 2 gives the rest point as it would be read with a stationary beam if friction were eliminated.

For example, in the following table we have:

	READINGS	
	Down	Up
(1)	[6.1]	
(2)		17.3
(3)	6.4	
(4)		17.0
Averages	6.4	17.2

where the first value 6.1 is not used except as a check on the second value just below. The readings occur in the order indicated by the numbers in parentheses. The rest point in this case is given by $(6.4 + 17.2) \div 2 = 11.8$, and the value 11.8 is the reading that the beam would assume if it came to rest at its position of equilibrium with effects of friction eliminated. This is illustrated in Fig. 2.

When the damping is large or for other cases where the taking of extraordinary pains is justified, the rest point can be determined by taking the average of a series of readings on one side and combining them with the average of a series of readings coming in between on the other. Thus, for illustration, in the following readings we have:

	READINGS	
	Down	Up
	6.1	17.3
	6.4	17.0
	6.7	16.7
	7.0	
Averages	6.6	17.0

The mean or rest point here $= (6.6 + 17.0) \div 2 = 11.8$.

11. ESTIMATION OF TENTHS

In the application of the method the tenths divisions are obtained by estimation. This can be done after very little experience. In the estimation of tenths there may be, of course, a difference of a tenth between two different observers. This, however, is of no importance if the sensibility of the scale is sufficient for the purposes of the test.

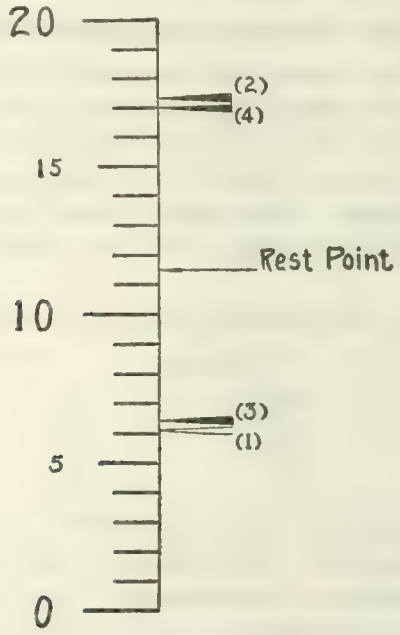

FIG. 2.—*Illustration of method for determining rest point from readings of the end positions of the pointer.*

Successive readings are indicated by (1), (2), (3), and (4). The rest point at 11.5 is determined from the average of readings (2) and (4) added to (3) and divided by 2. Reading (1) serves as a check on reading (3), but is not otherwise used in the computation.

12. UNNECESSARY TO DIVIDE BY TWO

In the preceding paragraph it has been mentioned that the rest point of the beam can be determined by adding the extreme readings, one in one direction and the other in the other, and dividing by 2. This would give the actual reading on the graduated scale that would be obtained if the beam were at rest. In practice, however, it is unnecessary to divide by 2, and in the form given later the nominal value of the rest point is taken as the sum of the 2 extreme readings. This saves time and introduces no errors.

The computations must be made consistent with the plan, of course, but this is easily done and without confusion. For instance, in place of the *sr* there is used one-half the *sr*. This is the change in weight on the platform that would change the sums of the extreme readings 1 division. The effect of not dividing by 2 is precisely the same as would be obtained if a graduated scale just twice as fine were used and the readings obtained on it were divided by 2.

13. MOUNTING OF POINTER ON THE TRIG-LOOP STAND

It is possible to place the pointer on the beam and the scale on the trig loop. In practice experience has shown that it is found to be of advantage to attach the pointer to the trig-loop stand and mount the graduated scale on the beam. This has the advantage that the level at which the readings are made does not vary, and it is not necessary to move the eye up and down to follow the pointer, and the illumination remains unchanged. In this case, however, care should be taken that the scale is placed and read so that the low readings correspond to the low position of the beam and high values for the high positions of the beam.

14. METHOD OF APPLYING THE WEIGHTS

In the test of a scale a known test load is applied on the platform and standard counterpoise weights are applied on the beam; the error in the scale is given by the lack of correspondence between the weights on the platform and the counterpoise weights on the beam. It has been found advantageous to use an even number of pounds, or some simple division of a pound, on the counterpoise and then make the necessary changes required to obtain the balance by changing weights on the platform. This method has many advantages, in that the use of weights of very small denomination on the counterpoise is avoided.

For convenience in carrying out the test and avoiding the use of small weights on the counterpoise it is the practice to balance the scale at zero load with a number of standard weights on the platform, varying from 1 to 50 pounds, according to circumstances. These should be greater than any error that is likely to develop in the scale. This scheme is of particular value in the case of scales which are fast.

For illustration, take a scale which is fast 1 pound per 1000 pounds, and let it be balanced so that the position of equilibrium of the beam is in the center of its play at zero load. If the nominal

multiplication is 1000:1 and a 10 000-pound weight is placed on the platform and a 10-pound weight is placed on the counterpoise of the beam, the scale will not balance, but the beam will be high. In order to obtain a balance, it will be necessary to remove 10 pounds from the platform or to add 0.01 pound to the counterpoise. It is desirable to avoid the use of small weights on the counterpoise; but in this case they would be needed, as it would not be possible to remove any of the solid 10000-pound weight placed on the platform. However, if the scale had been balanced with 20 pounds on the platform at zero load, then, when the 10000-pound weight was placed on the platform and the 10-pound weight was placed on the counterpoise, it would be possible and convenient to balance the scale by removing 10 pounds of the 20 pounds of weights which were placed on the platform at zero load. In this case it would not be necessary to use or to provide small weights for the counterpoise. Thus, where the scale is balanced at zero load with a few weights on the platform, weights of large denomination can be applied to the platform and the weights corresponding to the nominal amount required to balance can be placed on the counterpoise, and the balance can be determined by suitable changes in the small weights on the platform.

15. DESCRIPTION OF RECORD FORM AND COMPUTATION SHEET

With the preceding discussion in mind, the application of the method in connection with a record form and computation sheet used by the Bureau of Standards can be followed with understanding. Referring to Form 243 E, Fig. 3, the columns have been numbered in order.

Column No. 1 is for recording time. The time is not entered, as a rule, with each observation, but should be noted every few observations, particularly when the next step to be taken is different in character from what has just been carried out. The time record is of importance in some tests in correcting for a shift of the zero balance or other changes which appear to progress in a more or less uniform manner with time. This record is also, in practice, found to give information as to what time will be required to shift weights and conduct certain features of the test, so that it is possible to predict when the work will be completed and to make other deductions of practical interest in guiding the details of the work.

In column 2 are recorded the observation numbers. It should be noted that a number is assigned each time a change is made

or whenever the observations are interrupted. These numbers are of great value in referring to and identifying any of the data taken.

Column 3 is for the record of the counterpoise weights applied at the end of the beam. The values and identification of the

Form 347 B

Place Washington, DC Owner Bur. Standards Mr. _____ Date _____ Test No. _____ Page 11

MASTER SCALE TEST RECORD

BUREAU OF STANDARDS

DEPARTMENT OF COMMERCE

Weather Unclear Test

Wind _____ Temperature 70°F By A.S.B.

(1) Time	(2) Obs. No.	(3) Cliv. No. on end of beam	(4) Scale Readings Low	(5) Scale Readings High	(6) Scale Readings Sum	(7) 2D-Sum	(8) A. R. +2	(9) Corr.	(10) Pos	(11) Load	(12) B	(13) (1) Zero Load	(14) B + Corr. (2) Test Load	(15) Diff. (1)–(2)	(16)
PM															
6:15	76	10 lb 2	3.2	12.0	15.4	+4.6	338	+1.5	Gear	10.000#	20		21.5	+1.1	
			3.4	11.9											
			3.4	12.0											
	77	10 oz	—	+8.0	30.2	-10.2			10 oz	10 oz	2.5				
			12.3	17.8											
			12.4												
			12.4	17.8											
6:23	78	Gear	10+	16.4	26.6	-6.6	.368	-2.4	—	Gear	2.5	22.6			
			10.2	16.4											
	79	Gear	2.1	11.1	13.0	+7.0				Gear	2.0				
			2.0	10.9											
6:28	80	10 lb 2	4.6	11.6	16.2	+3.8	.331	+1.3	Gear	10.000#	2.0		21.3	+1.3	
			4.6	11.8											
	81	10 oz	10+	16.4	26.7	-6.7			10 oz	10 oz	2.5	Reject Anomalous			
			10.3	16.3											
6:31	82	10 oz	—	19+	31.3	-11.3			10 oz	10 oz	2.5				
			12.1	19.1											
			12.2												

FIG. 3.—*Example of record form and computation sheet showing the method of entering the observations and determining the errors in the multiplication of the scale*

weights can be entered here by using suitable abbreviations. Also, under usual conditions more than one line can be employed for this, as two lines or more are generally required for recording the readings of the pointer.

Columns 4 and 5 are for recording the readings of the pointer. When the first reading is a high one a dash should be made opposite in column 4 to indicate this fact. Otherwise, confusion exists; low readings which occurred after certain high readings appear to have come before them, and errors are introduced. In accordance with the requirement for a proper elimination of the damping, one observation on one side is used with the average of two observations on the other. Note that a line is drawn through the value not used or a circle is placed about it, so that there will be no misunderstanding on the part of the computer as to what was done or intended on returning to the work after the immediate recollection of the details have escaped the memory. This also indicates to the person checking the work just what was done.

In most cases, especially with observers who are particularly familiar with the work, it is not necessary to place the average values of the readings in columns 4 and 5, because these averages can be very readily computed mentally.

In column 6 is placed the sum of the average or representative values for the upper and lower extreme movements of the beam. If this were divided by 2 it would give the reading on the ruled scale which was the rest point of the beam. This division by 2, however, is unnecessary, as has been already explained.

If the graduated scale is adjusted so that the reading 10 is at the mid-point of the play of the beam and is at the point where the beam is in a horizontal position, then the value 20 in column 6 represents the mid-point or reference balance position of the beam. It may be thought of as the sum of 10 taken twice, once for the upper and once for the lower swing of the beam.

In column 7 is recorded the value obtained by subtracting the value in column 6 from 20.0. Therefore, if the value in column 6 is less than 20.0 the resulting value in column 7 will have a positive sign (+), and when it is greater than 20.0 it will have a minus sign (−). It is important that this matter of signs be given careful attention. A positive sign indicates that weight should be added to the platform to bring the beam to a position of equilibrium in the middle of the loop, and a minus sign indicates that weight should be removed to accomplish this.

It is strongly recommended that for saving time and eventual economy of effort the pointer and scale be adjusted so that the double value of the horizontal position of the beam be caused to read 20, or, if not 20, then 10, 30, or 40. A double reading of 20

can be used for all except unusual cases. When the double reading of the beam in its horizontal position is taken as other than 20, then that other value should be used in place of 20 in obtaining the values in column 7.

In column 8 the value of one-half the *sr* is recorded. This value is determined from entries usually made in column 16. The one-half *sr* is the pounds per division of column 7.

Experience indicates that for most purposes the *sr* should be determined for each particular load each time that load is applied and should be used for that particular load. The chief advantage of this is that it serves as a check on the accuracy of the readings taken on the beam. Also, after values of the *sr* are obtained for 2 or 3 loads a fair idea can be had of what is likely to be found for other loads, and this permits errors in readings or defective conditions to be recognized and corrected.

If, therefore, any mistake occurs in the readings determining the position of equilibrium, it will disclose itself in its effect on the value of the *sr* obtained. This is illustrated by observation 81 on the Form 243 E, where the *sr* did not agree with that of observation 77 and it was rejected. Agreement was reached on repeating. The difficulty was probably from reading the scale 5 divisions too low at one point.

In the test the same loads may be used time and time again, and many values for the *sr* for the load may be obtained. There is some argument for using one average value of the *sr* for all computations for this load. However, when this is done the data of the test are always held up, and no final figures are established before the test has come to an end. In most cases the practical value of using an average *sr* is not sufficient to justify the delays and corresponding difficulties thus introduced.

In column 9 is recorded the correction. It is the product of the value in column 7 and the corresponding value in column 8. It should carry the same algebraic sign as the value in column 7. It is the value, in pounds, which is required to be added to or removed from the load on the platform to bring the position of equilibrium of the beam to the center of its play in the loop.

In column 10 is recorded the position of the test load on the platform, together with information of interest in this connection.

In column 11 is placed the value of the test load applied. It will be observed that this is headed by "T + W." This is on account of the fact that this form was arranged primarily for use in connection with the test of master scales by the equipment of

the Bureau of Standards, and "T + W" stands for "truck and weights." In this column is recorded the test load or portion of the test load which nominally is required to balance the counterpoise weights on the beam. It does not include the smaller weights mentioned next and used on the platform to facilitate the conduct of the test.

Column 12, marked "B," is for the small weights which are applied on the scale platform as a matter of convenience in establishing the balance and determining the *sr*, as has been previously referred to. The letter "B" in the railroad track scale testing work signifies "Brass weights," as brass weights are generally used for this purpose. This column 12 represents the quantity of weights on the platform in addition to those which correspond to the counterpoise weights applied at the end of the beam.

Column 13 is used for data for zero load, and also, where the built-up test loads are used, for the balance taken for the lower load of the step before the standard test weight load is applied or after it has been removed. The values entered here represent the additional weight which would be required on the platform to bring the position of equilibrium of the beam to the middle of the loop. The entries are obtained from the values in column 12 plus those in column 9, all with due regard for sign. This is more fully explained in the illustration in the use of the form given later on.

Column 14 is similar to column 13, except that the data entered here represent results obtained when the test weights are applied to the platform.

In column 15 are recorded the differences in the values of columns 13 and 14, as indicated at the top. The values in this column represent the error in the scale—a plus (+) sign indicates that the scale is fast and a minus (−) sign indicates that it is slow. A scale is said to be fast when the scale indicates a change in load greater than the actual change made in the load on the platform, and it is said to be slow when the scale indicates a change in load less than actually takes place on the platform.

In column 16 are entered the values used in computing the value of one-half *sr* of column 8. The change in weight employed in determining the *sr* is entered here, together with the resulting change in double divisions produced in the rest point.

16. ROUNDING OFF OF DECIMAL PLACES

If the data are being recorded and computed to the tenth of a division, there will be times when a number results expressed to half a tenth, or 0.05. This occurs for the average of a sum in

which the last figure is odd. In this case the value is rounded off to the nearest tenth, and the value selected should be that one where the last figure is even. This is illustrated on the computation sheet in observation Nos. 76 and 77. This rule is one which is often used in scientific computations and has the advantage of being a fixed rule that can be applied without loss of time otherwise required for the computer to make up his mind what he will do, and work can be checked by different individuals who will make decisions alike. Moreover, in checking work it clearly fixes in many cases whether or not a minor mistake has been made in the computations.

17. DISCUSSION OF PARTICULAR EXAMPLE SHOWN ON THE FORM

Considering, now, the particular data shown in the illustration of the use of the record form in observation 76, a 10 000-pound test load was centered on the scale platform and a 10-pound weight was applied on the end of the beam to counterpoise. The nominal multiplication ratio of the scale was 1000:1. With 20 pounds of small brass weights on the platform in addition to the lot of 10 000-pound weights, the rest point or position of equilibrium of the beam was 4.6 divisions below the mid-position, as shown in column 7. (In this the double divisions are used, of course, thereby avoiding the need for dividing by 2.) On adding 5 pounds to the platform observation 77 was taken, and it was found that the rest point was 10.2 divisions above the mid-position of the beam, as shown in column 7. On comparing the rest point of observation 76 with that of observation 77 it appears that the change of 5 pounds on the platform changes the rest point of the beam 14.8 half divisions. This is shown in column 16. The one-half *sr* or pounds per half division is therefore $5 \div 14.8 = 0.338$. This is entered in column 8 opposite that value in column 6 which is nearest to the value 20.0, in this case observation 76. On multiplying the value of column 7 with that of column 8 it is found that a change of $+1.5$ pounds is required on the platform to bring the beam to a position of equilibrium in its mid-position. The plus sign indicates that weight should be added to the platform. Combining the value $+1.5$ in column 9 with the value of 20 pounds in column 12, there is obtained the value of 21.5, which is placed in column 14. Therefore, the beam will have a rest point in its horizontal position in the middle of its play in the loop, for the conditions of observation 76, if a load of 21.5 pounds is placed on the platform in addition to the test load of 10 000 pounds.

In observations 78 and 79 it is determined in a similar manner for zero test load that the small weights required on the platform to bring the beam to a position of equilibrium in the middle of its play are equal to 22.6 pounds. This is shown in column 13. From this result and that of column 14, observation 76, it appears that the error in the scale for a 10 000-pound load placed on the center of the platform is +1.1 pound, or the scale is 1.1 pound fast, as shown in column 15.

In observation 80 the test load of 10 000 pounds is reapplied, this time on corner 1. In observation 81, 5 pounds were added to determine the *sr*. However, it was found that the 5 pounds produced a change in the rest point of 10.5 half divisions, when in observation 77 with the same load 5 pounds produced a chaned in the rest point of 14.8 half divisions. The results, therefore, do not check. It appears that a mistake was probably made in observations 80 or 81. On repeating and obtaining observation 82 under the same conditions as 81, a different value resulted, and it was found that the *sr* of observations 80 and 82 now check with that of observations 76 and 77. Observation 81 was therefore rejected. There appears to be a difference of about 5 divisions between observations 81 and 82. This may have come from reading the pointer 5 divisions in error on one side.

This illustrates the value of taking the *sr* each time, as an error was detected and corrected. It can be understood, also, that if an average value of the *sr* were to be used, it would be necessary to wait until all of the observations had been made before entering new values in column 8, and this would hold up and prevent the entries in columns 9, 14, and 15.

From observation 80 and 82 the error for a 10 000-pound load placed on corner 1 is found to be 1.3 pounds. This happens to be in close agreement with the results obtained for the center position of the test load.

18. SUMMARY

In conclusion it may be remarked that the method of test outlined herein gives a systematic and scientific procedure for making tests of unusual accuracy on compound lever scales. This method has been developed in the testing of railroad master scales, hopper scales, and other scales, and is found to be satisfactory. In the interest of uniform and efficient methods it is recommended and outlined here for those who may desire to adopt the procedure.

WASHINGTON, March 10, 1921.

DEPARTMENT OF COMMERCE

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BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 200

AN INVESTIGATION OF OXYACETYLENE WELDING AND CUTTING BLOW-PIPES, WITH ESPECIAL REFERENCE TO THEIR DESIGN, SAFETY, AND ECONOMY IN OPERATION

BY

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AN INVESTIGATION OF OXYACETYLENE WELDING AND CUTTING BLOWPIPES, WITH ESPECIAL REFERENCE TO THEIR DESIGN, SAFETY, AND ECONOMY IN OPERATION [1]

By Robert S. Johnston

ABSTRACT

Apparatus from 14 of the most prominent manufacturers were tested under standardized conditions, the practical work of cutting and welding being carried on by a group of experienced welders and cutters directly under the supervision of the Bureau representative.

None of the commercial cutting blowpipes procurable appear to be designed according to definite theory. None of the cutting blowpipes are efficient in cutting metal of all thicknesses. Considerable improvement can be made in economy in cutting 2-inch metal and possibly in other thicknesses. About 12 inches is probably the maximum thickness that may be cut economically with oxyacetylene blowpipes.

None of the welding blowpipes were correctly designed, and none were free from flash-back phenomena. Most of them are somewhat unsafe and inherent defects in design result in unsound welds. With a properly designed welding blowpipe, it is believed that satisfactory fusion welds may be made.

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[1] The paper presented herewith is the result of a special investigation conducted at the Bureau of Standards for the War Department, as represented by Maj. A. B. Quinton, jr., tank, tractor, and trailer division, Ordnance Department, through which department the results of the investigation have been made available for public distribution.

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I. INTRODUCTION

1. PURPOSE OF THE INVESTIGATION

During the war period oxyacetylene welding and cutting equipment became of primary importance to the American Expeditionary Forces as a means of demolition or field repair of various engineering structures. The increased use to which this apparatus was put necessitated the purchase by the Government of a large number of sets of welding and cutting equipment. Investigation showed that there were no reliable data available upon which specifications for this equipment could be based.

After several conversations between representatives of the War Department, members of the Bureau staff and representatives of the oxyacetylene blowpipe manufacturers, the Bureau of Standards received under date of September 14, 1918, over the signature of Capt. H. Carlton, miscellaneous section of the Ordnance Department, a letter requesting, in the name of the Chief of Ordnance, that the Bureau of Standards make a test to determine the "efficiency, safety, and workmanship entering into the several makes of apparatus" (oxyacetylene).

2. HISTORY OF DEVELOPMENT OF THE TESTS

In compliance with this request, a preliminary investigation was inaugurated in cooperation with various divisions of the Army and with the assistance of blowpipe manufacturers to plan tests which should indicate the relative merits of various blowpipes. The investigation originally contemplated an immediate testing of equipment and therefore would have been confined to a short series of rough tests. The armistice removed the urgency of the investigation and the rapidly changing personnel delayed it still further. The delay, however, allowed the scope of the investigation to be extended, and finally a thorough and comprehensive series of tests was developed.

The tests were finally specified after consideration of all suggestions from the various divisions of the Army and Navy and from the manufacturers who submitted apparatus for test. In planning the tests, care was taken to eliminate, as far as this was possible, any discrepancies due to the personal equation of the operators.

In their final form the tests were approved by a majority of the manufacturers, but two manufacturers were dissatisfied and withdrew the apparatus they had submitted for test. In these cases their apparatus was purchased in the open market and tested under the same conditions as the rest. When the schedule of tests had been prepared, a preliminary series of tests was run and the test procedure modified wherever it seemed advisable. The final schedule of tests which was then sent to the manufacturers was adhered to throughout the whole series. The conditions under which the manufacturers submitted their apparatus are given below.

3. STIPULATIONS GOVERNING THE CONDUCT OF TESTS

The stipulations forwarded to the manufacturers were as follows:

The investigation will include:

1. Tests of welding torches.
2. Tests of cutting torches.
3. Tests of regulators.
4. Tests of oxyacetylene welded specimens.

Throughout this series of tests the following conditions will be adhered to:

Gas:

1. All gas used will be of the same analysis within commercial limits.
2. Throughout the series of tests the gas consumed will be drawn from a bank of tanks so that its properties may be as uniform as possible.
3. Identical equipment in the gas lines will be used in all tests.

Torches:

4. The torches will be operated throughout the series of tests in accord with the instructions of the manufacturers.
5. The torches will be operated throughout the series of tests by one set of experienced welders and cutters who will work under the direction of the Bureau's representative in charge of the tests, except that a manufacturer may make a demonstration test using an operator of his own selection.
6. All makes of welding torches will be put through an identical series of tests. The gas pressures will be controlled by one set of gas regulators selected for this purpose.
7. All makes of cutting torches will be put through an identical series of tests. The gas pressures will be controlled by one set of gas regulators selected for the purpose.
8. All cutting tests will be made by mechanical guidance and control of the cutting torch.

Test material:

9. The material used for test purposes consists of ¼ inch, 2-inch, and 6-inch steel, each of known chemical analysis.
10. One grade of welding rod of known chemical analysis will be used for the welding tests.

Submittal of apparatus:

11. Any apparatus not voluntarily submitted for test by the manufacturer may be purchased in the open market and tested.

Information concerning tests:

12. Information concerning the tests to which the torches are to be subjected will be made available to the manufacturer before the actual testing is started. All apparatus voluntarily submitted for test must be in the Bureau's hands before the information concerning the tests is given out.
13. A copy of all data obtained during the test of a particular make of apparatus will be furnished the manufacturer of that apparatus.

Witnessing tests:

14. The tests on any particular make of apparatus may be witnessed by three representatives of the manufacturer of that apparatus.
15. Whenever the manufacturer's representatives are present they shall specify in writing that the apparatus tested is of their own manufacture, that it is identical in all respects with the torches of similar style sold under their name in the open market, that it was in proper working order at the beginning of the test (it shall be tried out by a competent representative of the manufacturer before this statement is made), and that the tests were carried out in accord with the schedule made available to them.

Retest and demonstration:

16. Any objections to the methods and procedure in operating the torches during the tests shall be submitted in writing within 24 hours after the completion of the test on any make of torch.
17. When conditions seem to warrant it, the Bureau may make such retests as it deems advisable or necessary.
18. When in the opinion of the manufacturer of a torch he can demonstrate that by the use of his own operators he can produce better results than were obtained by the Bureau of Standards' operators, such a demonstration test will be permitted, provided the manufacturer supplies the material to be used and further, holds himself responsible for all cost necessary in connection with the making of a demonstration test.

Retest and demonstration—Continued.

19. If the opportunity to make a demonstration test is accepted by a manufacturer, such test must be made at the Bureau under the identical conditions, except for operators, in which the Bureau's test was made. Further, the same torches and identical material are to be used for both tests.

20. The Bureau will endeavor as far as possible to assist the manufacturer to secure material of identical composition for demonstration tests.

Decisions:

21. Final decision upon any questions which may arise in regard to these tests will be made by a committee composed of one representative of the U. S. Army, one representative of the U. S. Navy, and three representatives of the Bureau of Standards.

4. ACKNOWLEDGMENTS

In connection with the development and completion of this investigation, the general experience of a number of people was freely drawn upon so that the question might be viewed from its broadest aspects, and such assistance is here acknowledged with the realization that the successful completion of the investigation was largely due to the splendid cooperation received. In this connection the assistance of the following is acknowledged:

The Ordnance Department of the War Department, U. S. A.; U. S. Naval Gun Factory, Washington, D. C.; Hull Division, U. S. Navy Yard, New York; and various manufacturers of oxyacetylene equipment, especially the Davis-Bournonville Co. and the Commercial Acetylene Co. Of the Bureau staff, G. M. Deming and his assistant, L. R. Sweetman, to whom the successful design, calibration, and operation of the testing equipment are largely due, and S. W. Miller, consulting engineer, Rochester, N. Y.

II. DESCRIPTION OF EQUIPMENT USED IN TESTING THE BLOWPIPES

In general the equipment used for making the tests may be listed as—

1. The weighing system for determining the amount of the gases used during the tests by loss of tank weight.
2. The gage-board system containing the necessary pressure gages, regulators, and orifice flow meters.
3. The welding table.
4. The cutting table.
5. The flash-back and safety testing apparatus.

A general view of the entire equipment and its arrangement is shown in Fig. 1. For purposes of analysis and explanation it

may be more easily studied by reference to a diagrammatical sketch, Fig. 2.

Reference to Figs. 1 and 2 will show that the tanked gases were "banked" and suspended by means of a hanging platform from

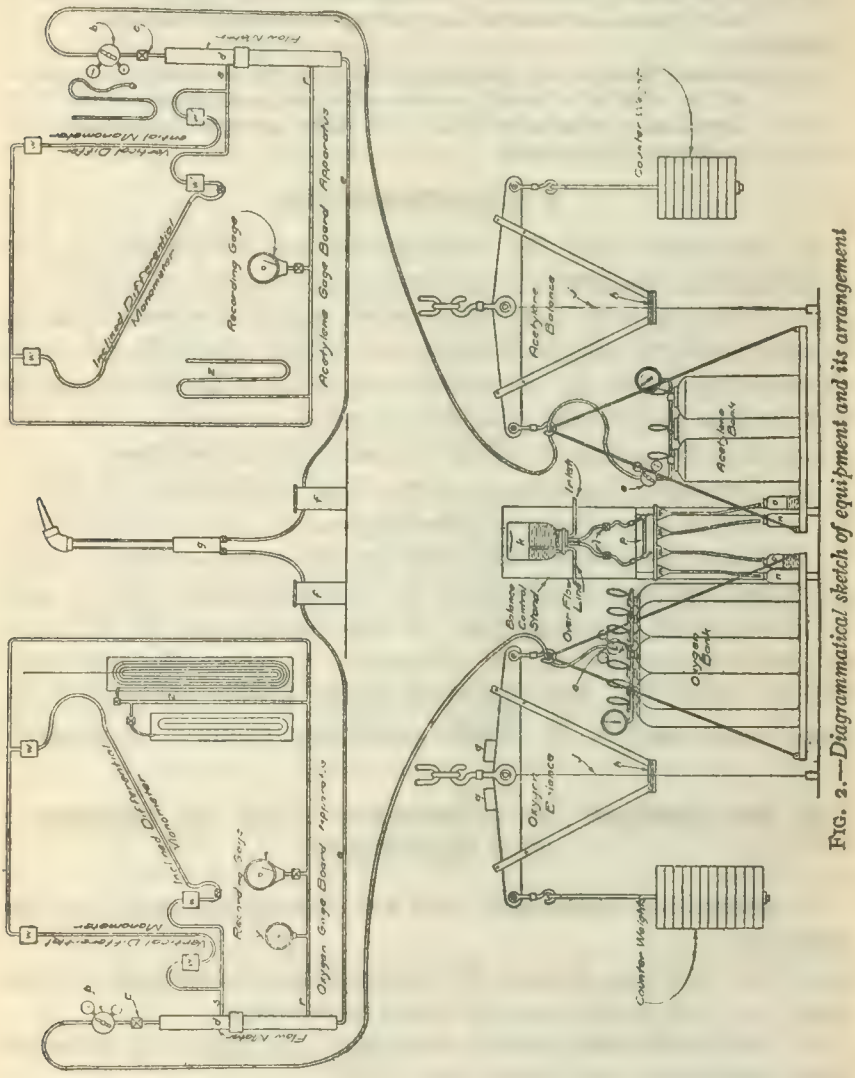


FIG. 2.—*Diagrammatical sketch of equipment and its arrangement*

one arm of an equal arm balance, their weight being counterpoised by dead weights suspended from the other balance arm.

The gas from the tanks on the balance passed through a regulator (*a*) thence through a flexible hose to the back of the gage board. Passing through the gage board the supply line entered a second regulator (*b*) thence through a needle valve (*c*) to the

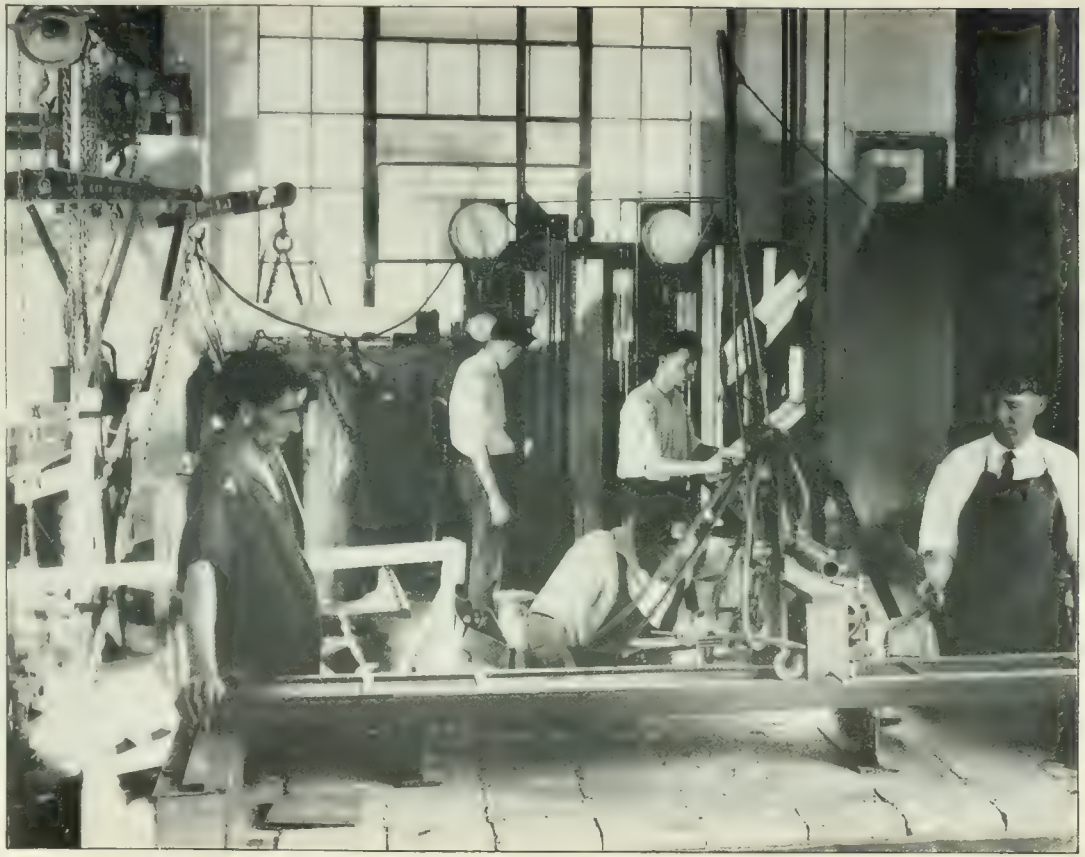


FIG. 1. *General view of equipment and its arrangement*

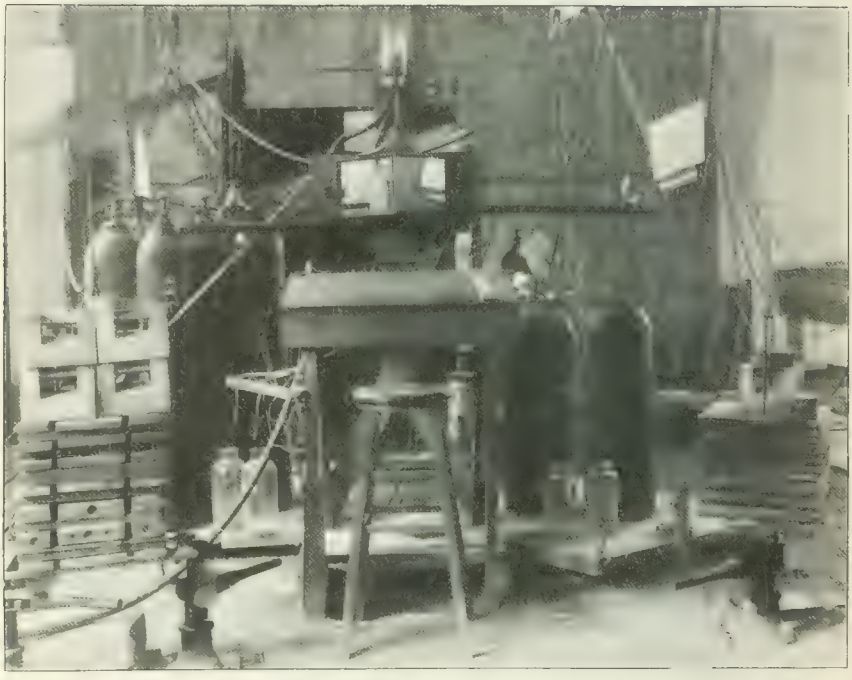


FIG. 5.—*Weighing apparatus showing banked gas tanks on balances and observer's table*

top of and through an orifice flow meter (*d*). The gas coming from the extreme bottom of the flow meter was then conducted through a standardized length of flexible hose (*e*) containing a safety flash-back tank (*f*) to the blowpipe to be tested.

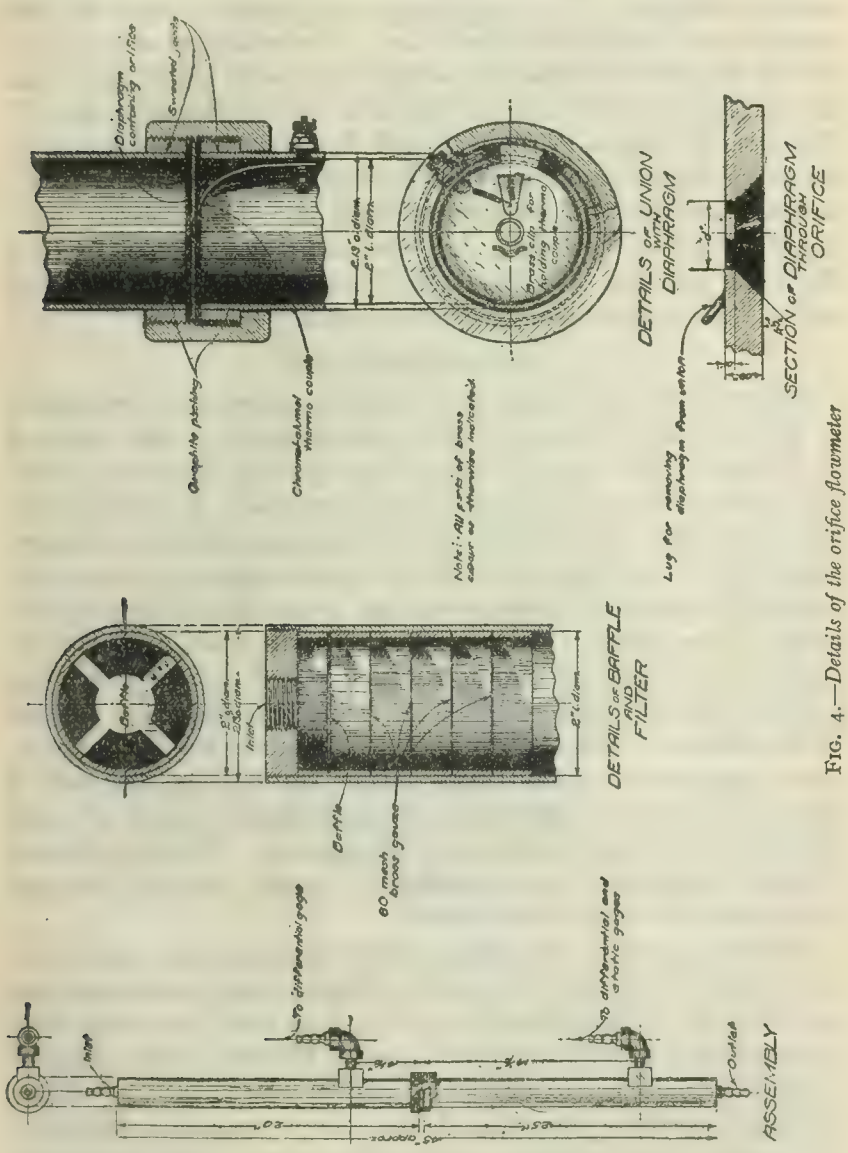


FIG. 4.—*Details of the orifice flowmeter*

1. THE WEIGHING SYSTEM

The weighing system comprised two equal-arm balances of 3000 pounds and 1000 pounds capacity, respectively, for the oxygen and acetylene and the observer's table. (See Fig. 3.)

(*a*) THE BALANCES

The "banked" tanked gas was set on a platform suspended from one arm of the balance. Dead-weights were hung from the other arm of the balance to counterweight the load of the suspended platform. The weighings were always made by the substitution method described later. The readings were secured by the so-called "Method of swings" and were taken on a scale (*h*) as it oscillated to either side of a fixed reference point (*j*). The scale (*h*) was a centimeter scale fastened to a support hanging down below the beam of the balance and rigidly fixed to the balance beam so that it oscillated with it. The necessary fixed point (*j*) was secured by a fine wire stretched taut in front of the scale between the center knife-edge support of the beam and a stake driven in the ground beneath.

(*b*) THE OBSERVER'S TABLE

Between the two balances, with arms projecting horizontally to within a short distance of and at right angles to the plane of oscillation of the balance beams there was an observer's table. The extreme ends of the projecting arms held a small electric light bulb and a magnifying glass which permitted an enlarged projection of the scales to be thrown on two mirrors set above the desk of the observer's table and at 45° to the line of the projected image of the scales. This arrangement permitted the observer, seated at the desk, to obtain a clear view of the scale of either balance instantaneously. Directly below the reflecting mirrors of the observer's table, and within convenient reach of the observer, there was placed a small tilting holder (*p*) carrying the ends of soft rubber tubes connected to a constant level reservoir of water (*k*) placed above and to the left of the mirrors. The flow of water through these tubes was controlled by needle valves (*l*) located to the right and left of the mirrors.

The tube ends on the tilting holder deposited the water from the reservoir into one of two pairs of funnels (*m*) so arranged that by tilting the holder either forward or backward the water could be deposited into either pair of funnels. Flexible tubing attached to these funnels led to bottles (*n* and *o*) placed on the balance platforms carrying the tanked gas. It is apparent, therefore, that water could be easily deposited in either of the two bottles on the balance platforms by simply tilting the tube holder backward or forward. Control of the rate of flow of the water was readily governed by the needle valves (*l*). An observer sitting at the desk and watching the swing of the balance arms could by

simple adjustment of the needle valves permit water to flow into the bottles on the balance platforms at such a rate as would approximately compensate for any loss in weight due to the withdrawal of gas from the tanked cylinders.

(c) METHODS OF DETERMINING WEIGHT DATA

With the necessity of using gas until the blowpipes were properly adjusted before commencing any test operation and because there were two balances to observe as nearly simultaneously as possible at the start of each test, it was not possible to always start the tests with the balance scales at an exact zero. There was a constant and regular "drift" of the zero position as gas was withdrawn from the tanks. Any error from this possible source was readily and easily compensated for as follows: After the blowpipe was properly adjusted and the gage-board observers and practical operators were ready to begin a test, the weighing observer was notified by signal. The latter meanwhile had been adjusting the flow of water to such rate as would as nearly as possible compensate for the weight of gases being withdrawn. At the sounding of the signal he began a series of five turning-point observations for each balance, the five for the oxygen balance being taken first. At the completion of the fourth turning-point observation for the oxygen balance a preliminary warning signal was given, and at the completion of the fifth observation the signal to begin the test was sounded. Immediately following this signal the weighing observer took the five turning-point observations for the acetylene.

The mean of each consecutive three turning-point observations for each gas gave three determinations of the zero of the balance. From these the "drift" of the zero of the balances could be determined and the readings corrected to the actual time of starting the test.

During a test the zero of the balances was maintained as nearly constant as possible. The exact value could not, however, be held for so long a period as the tests required, so that at the completion of a test the zero usually had shifted to one side or the other of the initial zero determination, because either too much or not enough water had been permitted to enter the gas weight bottles on the scale platforms. Such errors were readily corrected, however, by the determination of the final zero.

In order to make this correction it was necessary to determine the sensitivity of the balance. As this did not remain constant

from day to day, the sensitivity was determined each morning before beginning the day's tests. It was desirable to release the load on the balances each evening, but a complete release of the load was found to cause erratic changes in the sensitivity. To avoid this the greater portion of the load was relieved by jacks placed under the counterweights, allowing enough weight to remain to prevent displacement of the knife-edges. With this procedure erratic results from the determinations for sensitivity were very infrequent.

The sensitivity was determined by taking turning-point observations for the zero of the balance oscillations and then determining by the same method the "drift" caused by placing a standard 10 g weight on the counterweight side of the balance. The amount of drift in centimeters divided by the number of grams used to cause such displacement gave the weight equivalent value of the scale divisions in grams per centimeter. This determination was always made with the gas line leading to the gage board "flooded" under a pressure equivalent to the average gas pressure generally used; that is, 125 lbs./ in.^{2} for the oxygen line and 20 lbs./ in.^{2} for the acetylene line.

During the course of the experiments it was thought desirable also to make one other correction for error in balance determinations. It is almost impossible to stop all leakage when a series of tanks is banked as in these tests. Small leakages in tank valves, too small to be readily detected, were found to exist. To compensate for these the gas lines were flooded as far as the regulator on the gage board and the zero-point determinations made over a period approximating a half hour. The determination of the amount of drift thus found in connection with the weight equivalent value of the scale divisions gave an index of the rate of loss of gas by leakage. This determination allowed all balance data to be corrected for such loss. (See Fig. 19.) If the gas loss exceeded 0.01 to 0.02 pound per hour, actual testing of the blowpipes was not continued until the leaks causing such losses were located and stopped.

(*d*) PRECISION OF WEIGHINGS

As to the precision of readings obtained by the use of these balances and the weighing system it would seem that, giving due consideration to all disturbing influences, the values obtained for gas consumption are accurate to 0.005 pound, and in most cases are probably much closer.

2. THE GAGE-BOARD SYSTEM

(a) GAGE-BOARD REGULATION

During the preliminary investigation of the equipment to be used for these tests it was found that none of the standard regulators generally furnished with such equipment were satisfactory for regulation purposes, if results of any considerable accuracy were to be obtained. Due to the construction of the ordinary regulator the readjustments for pressure distribution are intermittent and spasmodic and allow of considerable variation in the actual pressures delivered on the low-pressure side of the regulator.

It became necessary, therefore, to devise some means by which a more constant and reliable pressure could be delivered to the line supplying the blowpipes. After considerable inquiry and experimental trial the method shown in Fig. 2 and the various photographs was adopted and as a whole proved extremely satisfactory. The main regulation was accomplished by two regulators used in series, one attached to the outlet of the manifold of the banked tanks (*a*, Fig. 2), and the other placed on the gage board just in front of the inlet to the flow meter (*b*). By means of the regulator at the manifold the tank pressures were reduced to an amount about 50 per cent in excess of the pressures required for the blowpipe, the second regulator being used to reduce still further the pressure to that desired.

The various regulators used were the best of a large group that was on hand, their efficiency being determined by constant trials and observations of the regulators themselves. Through this means a set of regulators selected for their good performance was used as a whole throughout the entire series of tests.

The regulator (*b*) at the gage board was changed as occasion demanded in order to secure as sensitive a regulator as possible contingent upon the pressure which it was required to deliver.

It was found, however, that even with the selection of regulators from the large number on hand, too great a variation still existed in the pressures which would have been delivered to the blowpipe line. A part of this trouble was due to the construction of the regulator, in that the reduction of pressures was accomplished by intermittent and spasmodic action of the regulator. Recognizing this, a set of "rappers" was used which vibrated the body of the gage-board regulator (*b*), the idea being that the constant agitation and vibration of the regulator body would tend to keep

the seat of the regulator floating and therefore secure less intermittent action. These "rappers" were connected in series with a master vibrator located at some distance from the board. The interrupting mechanism of the two gage-board vibrators was short-circuited so that no sparking could be produced at the gage board within a dangerous vicinity of possible gas leaks, etc.

Further experience indicated that the insertion of a small hand-regulated needle valve (*c*) between the regulator at the gage board (*b*) and the flow meter (*d*) secured a decidedly more uniform regulation. The standard procedure throughout the tests, therefore, was to have one of the gage board operators make such adjustment by means of the needle valve (*c*) as from time to time was found necessary to maintain constant pressures in the gas line. After a blowpipe had been properly adjusted and had been operating for a few minutes it was seldom found necessary to make any further adjustment in the needle valve, the pressures remaining as a whole very constant through the tests.

By means of the above arrangement it was readily possible to maintain under most all conditions a pressure varying not more than 0.01 or 0.02 of a pound from the desired amount. The constancy of the regulation obtained may be seen from Fig. 21, which is a copy of an average log sheet, or from Fig. 31, a copy of one of the autographic pressure curves. As a general thing the best regulation obtained was far better than that shown on these figures, although occasionally somewhat poorer regulation resulted.

(*b*) FLOW METER

During the study of the methods proposed for testing the various blowpipes it appeared desirable to secure information concerning the amount of gas used through some independent means other than through the loss of weight of the tanks. It seemed essential that these data should be secured as a check upon possible errors in tank weights and further to permit of instantaneous readings from time to time of the relative amounts of gas being consumed by the blowpipes.

After considerable discussion and study it was finally decided that this information could be best secured through the use of a flow meter. A special orifice flow meter, the general details of which are indicated in Fig. 4, was therefore designed and constructed for this investigation.

Referring to Fig. 4 it will be seen that the flow meter consisted essentially of a brass tube, 45 inches in length and 2 inches inside diameter, which was cut into two pieces at approximately the center of its length and fastened together by a screwed coupling. To the top and bottom ends respectively of the tube in line with its longitudinal axis, an inlet and an outlet nipple were brazed. Two other nipples were brazed to the side of the tube, one a short distance above the split joint at the center of the tube, and the other about 10 inches above the outlet nipple. These led respectively to the differential manometer and to the differential and static pressure manometers.

In order to insure uniform distribution of gas throughout the inlet end of the flow meter there was built into the upper end just below the inlet nipple a cross baffle and a series of four 60-mesh brass screens, as indicated in Fig. 4.

The orifices were of the thin plate, sharp-edged type, and were held in position at approximately the mid-length of the flow-meter tube by the adjustment of the coupling as indicated in Fig. 4. A section showing their shape is also included in this figure. Orifices of this type were selected as they were simple and readily machined to exact sizes. Their continued use throughout the series of tests has shown that they have been extremely satisfactory. A total flow of several hundred million linear feet of oxygen has passed through one of the orifices used on the oxygen flow meter at an average velocity of several thousand feet per minute, and so far as can be detected it has shown no signs of wear or erosion. To substantiate this latter a photograph was taken of this orifice before it was used and after it had been put through the entire series of tests. This photograph does not even indicate any rounding of the corners of the orifice, and the check measurements show also that no enlargement of the opening has taken place.

Experience showed that the number of orifices could be limited to five without undue loss of sensitivity by using, in parallel with the main vertical manometers, adjustable inclined manometers. These water manometers had a relatively high sensitivity and were therefore protected by cut-off valves against excess pressure. (See Figs. 1 and 7.)

Since the rate of flow of a gas is a function of its temperature as well as its pressure and density, it was necessary to measure the temperature of the gas as it passed through the flow meter. In order to accomplish this a thermocouple was built into the

flow meter, its terminals being placed directly below the tube coupling. That it might furnish a reliable index of the temperature of the gas and at the same time be free from any possible effects due to the impingement of the gas upon the thermocouple itself, it was held firmly in place just inside the orifice opening by a spring clip as indicated in Fig. 4. Further, that the temperature readings might be affected as little as possible by disturbing interferences from the outside atmosphere, the flow meters were carefully insulated with wool felt lagging and the oxygen meter was still further protected by being incased in a wooden box. A copper-constantan thermocouple was used for the oxygen meter, and a chromel-alumel thermocouple was used for the acetylene meter.

The thermocouples led to a potentiometer and were connected to a cold junction placed in an ice bath in a vacuum tube.

Flow-Meter Theory.—The approximate theoretical formula upon which the use of the flow meter was based is—

$V = K\sqrt{\frac{P_a h}{T_a G}}$, in which

V = the rate of flow of the gas in cubic feet per hour at 760 mm mercury pressure and 32° F temperature;

K = the orifice constant;

P_a = absolute pressure corresponding to $P = P + B_p$;

P = the flow-meter static pressure (as measured at r, Fig. 2);

B_p = atmospheric pressure;

h = flow-meter differential as read in centimeters of water;

T_a = absolute temperature;

G = density of the gas.

When the gas is a simple gas of constant density, such as oxygen, the formula may be more conveniently used in the simplified form $V = K' f\sqrt{P_a h}$ where $f = \sqrt{\frac{T_m}{T_a}}$ and $K' = \frac{K}{\sqrt{G}}$, T_m being a standard mean temperature.

The value f generally will never vary greatly from unity, as T_a and T_m are as a rule nearly equal. It is therefore possible to ignore the factor f in approximate computations, with resulting errors probably not greater than 2 per cent.

These formulas can readily be modified so that it is not essential that T_a be measured on any specific temperature scale. This allowed the use of the thermocouple electromotive forces themselves as determined by a potentiometer. For convenience, the relation between the electromotive forces and the oxygen-temperature factor shown in Fig. 5 was developed.

Where the gas is a complex one of changing density such as that drawn from compressed acetylene cylinders which contain varying proportions of acetone, the fundamental flow-meter formula may be written, for convenience, $V = k'F\sqrt{P_a h}$, in which $k' = \frac{K}{\sqrt{G_m}}$, G_m being a standard mean density, and $F = \sqrt{\frac{T_m G_m}{T_a G}}$.

G_m was taken as 0.0750 lb./ft.^{3}, which appeared to be the mean density of the gas as taken from the density determinations mentioned later. The value selected for T_a was identical with that used for the oxygen meter. In this later form the temperature density factor F also ranges around unity, so that, as indicated above, it might also be neglected in approximate determinations. In such cases, however, the results will be more liable to error on account of the varying density of the acetylene. The relations between the density, gas temperature, and temperature density factor F, which were used in the interpretation of the test data, are shown in Fig. 6.

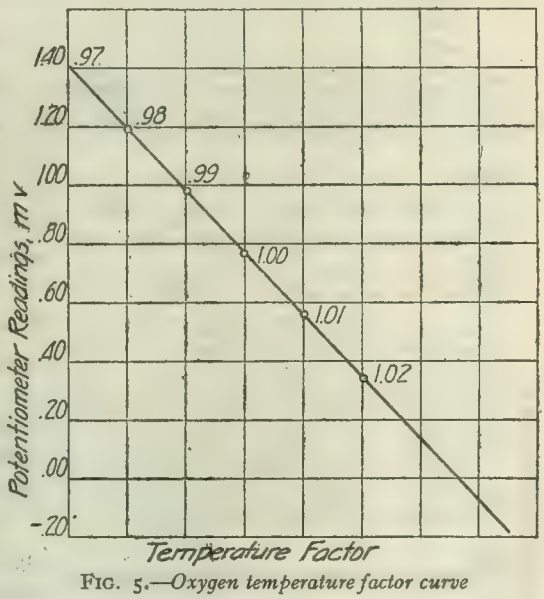


FIG. 5.—*Oxygen temperature factor curve*

CALIBRATION OF FLOW METERS.—Three independent calibrations were made of the orifices of each of the flow meters to determine the value of the constant K. In the first calibration the flow-meter readings were compared with the readings of a standardized dry gas meter. The pressure regulation was not as close as was desired, and the values of K fluctuated more widely than was considered satisfactory.

A second calibration was therefore made after the selected regulators and "rappers" were installed in the form used in the final blowpipe tests. As an additional check the dry gas meter readings were supplemented by the weight of the gas lost as determined by the balance. The regulation secured was excel-

lent and as a whole far better than was obtained in the blowpipe tests themselves. The results of this calibration were very consistent, as may be seen from the following table of values of K determined for one orifice under three widely different pressures and three widely different rates of flow:

Test No.	K
o–1	1.171
o–2	1.169
o–3	1.182
o–4	1.187
o–5	1.190
o–6	1.187
o–7	1.186

This second calibration was therefore at first considered satisfactory. In computing the results of the first blowpipe tests, there appeared an unexplained systematic difference between the gas consumption calculated from the balances and from the flow meters. Investigation showed the unexpected result that the value of K, as determined by the manometers used, was lowered by pressure fluctuations.

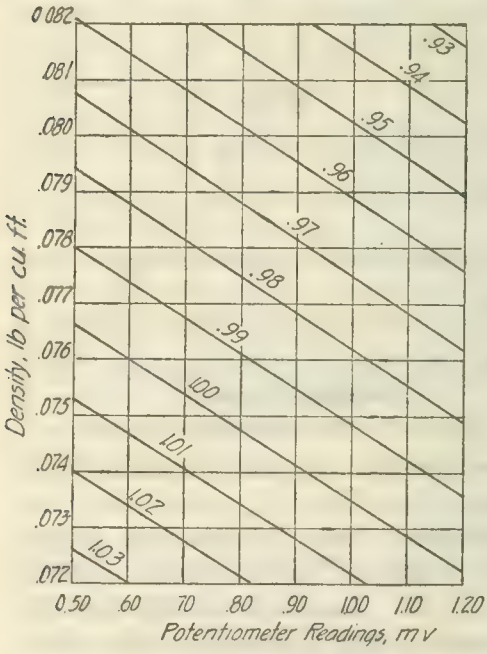


FIG. 6.—*Acetylene temperature factor curves*

The theoretically existing difference between the square root of the average value of h and the average of the square root of h is far too small to account for this difference. It must therefore be due to an actual higher average differential under fluctuating pressures, due to inertial and frictional effects in the manometers, or to a systematic tendency of the operator to overread a fluctuating manometer.

A full investigation of the cause of this systematic difference would be very desirable, but the time required would have been excessive in the present investigation.

A third calibration of the flow meters was therefore calculated from the balance and flow-meter readings in the preliminary

series of actual blowpipe tests. The pressure regulation in these tests was not so good as in the second calibration but better than in the first. The magnitude of this systematic effect of pressure fluctuation can be seen from the following table:

Calibration	Pressure regulation for orifice No. 5	K
2	Very good	1.189
3	Average	1.145
1	Poor	1.114

The effect is larger in the smaller orifices.

The flow-meter constants K, as determined from this third calibration, were used in computing the test data. The close check of the gas consumption calculated from the flow-meter readings with the balance readings in the individual tests (see for instance Figs. 22 and 23) shows that this was justified, and indicates that the same average closeness of pressure regulation was maintained throughout the tests.

(*c*) MANOMETER AND PRESSURE-GAGE EQUIPMENT

As indicated above, the gas leaving the flow meters passed to the blowpipe through the gas line *e* and flashback *f*, Fig. 2. The static pressure of the gas passed to the blowpipe was determined by means of pressure gages and manometers on the line *r*, Fig. 2, taken from the side of the flow-meter base just above the outlet to the gas line *e*.

On the oxygen side of the gage board there was incorporated in the line of static pressure *r*, Fig. 2 or 7, a Bourdon tube master gage *y*, a recording gage, and two compound manometers *z*, one of 0–44 lbs./in.^{2} range, and the other 0–180 lbs./in.^{2} This static line pressure also connected through the baffles *w*, Fig. 2, with the upper ends of the differential manometers. The other ends of the differential manometers, both vertical and inclined, were connected through reservoirs *w'*, Fig. 2, to the static pressure lines at the inlet end of the flow meter just above the orifice opening.

On the acetylene side of the gage board the Bourdon tube gage was replaced by a simple mercury manometer *z'*, which could be connected into the line as needed, and in place of the compound manometer *z* of the oxygen side there was installed a simple U-column mercury manometer.

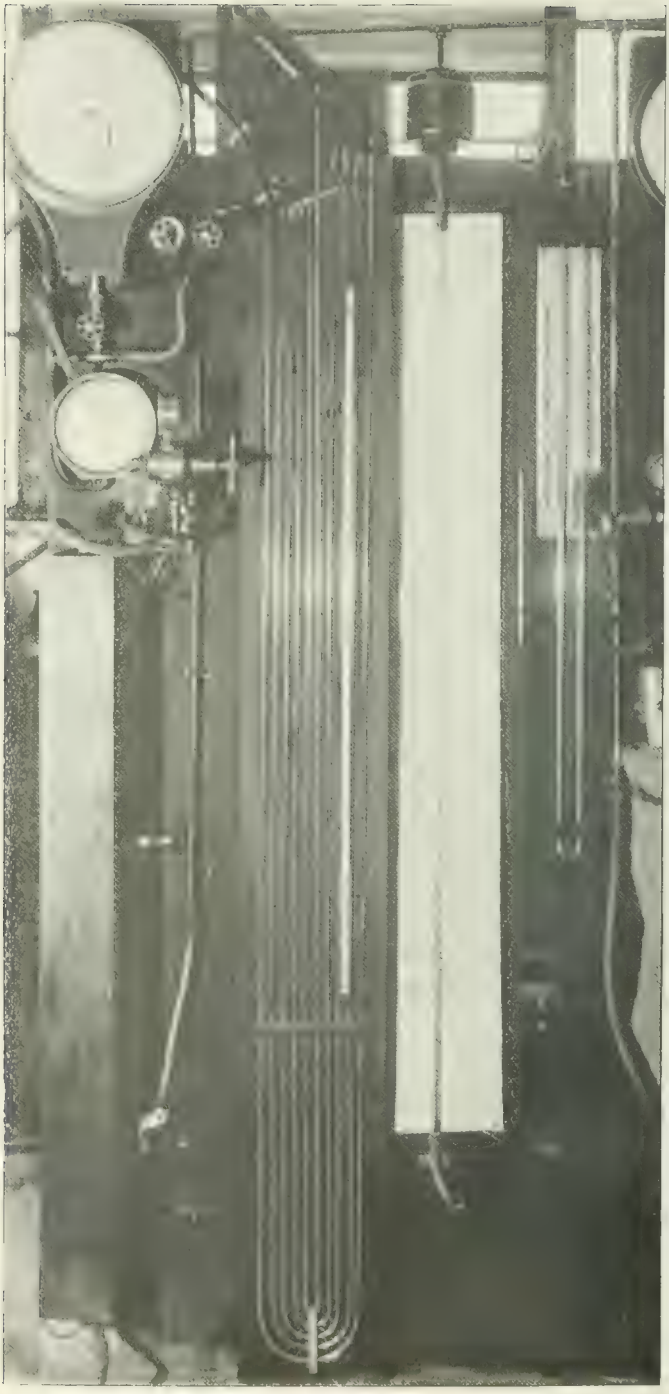
The pressure gages used throughout the tests were a special series of master test gages that were made for this investigation.

The one at the oxygen side of the gage board was of 0–300 lbs./in.^{2} range, the one at the acetylene manifold of 0–500 lbs./in.^{2} range, and the one on the oxygen manifold of 0–3000 lbs./in.^{2} range. These gages were very carefully calibrated before being used for test purposes, and their calibration checked during the tests and at the completion of the tests. The Bourdon tube gages at the oxygen and acetylene manifold were used throughout the entire series of tests. On the other hand, the Bourdon tube gage on the oxygen side of the gage board *y*, Fig. 2, was seldom used, except in cases of heavy cutting where the pressure desired for delivery to the blowpipe exceeded the 180 lbs./in.^{2} range of the large compound manometer. The pressures actually delivered to the blowpipe, except as noted above, were always maintained constant by observation of the mercury manometers.

It was felt desirable to have an autographic record of the pressures at which the gas was delivered to the blowpipes for the various tests. Accordingly two special recording gages were installed in the static pressure line, the one used on the oxygen line having a range of 0–50 lbs./in.^{2}, and the one on the acetylene line a range of 0–15 lbs./in.^{2} A copy of one of these autographic records is shown in Fig. 31, which will give visual evidence of the uniformity of the pressure under which the blowpipes were operated during the tests.

As indicated above, the pressures during the greater part of the tests were maintained by observation of mercury manometers. Those on the oxygen side of the line are of particular interest in that they were special compound manometers devised for this investigation. In principle they consisted of a series of mercury U-tube manometers connected at their top by a water column in such a way that the movement of the first mercury column, in response to pressure from the line *r*, was transmitted to the second mercury column and simultaneously to the remaining mercury columns through the agency of the intermediate water columns. Fig. 8 shows the general construction of the larger of these manometers, which is a five U-tube manometer having a range of 0–180 lbs./ in.^{2} The range of this manometer was increased by the addition of an upright riser filled with mercury, which displaced the initial zeros of the U-tube enough to increase the capacity of the manometer to an amount equal to the weight of the extra added mercury column. This instrument was used for maintaining oxygen pressures in excess of 50 lbs./in.^{2} On account of its extreme range the scale subdivisions indicating

FIG. 7.—*Gage board with its equipment, showing gages and manometers*



pressures lower than 50 lbs./ in.^{2} were too small to permit of ready interpolation of pressures to the nearest 1/100 of a pound. There was therefore used in connection with this a smaller compound manometer of two U-tube construction with a range of 0–50 lbs./in.^{2} This latter was used for all welding tests and for the larger number of cutting tests.

These manometers gave extremely satisfactory service and proved a valuable addition to the gage-board equipment. The smaller compound manometer of 0–50 lbs./ in.^{2} range proved to be extremely accurate and sensitive, and it was therefore selected as the pressure standard for the entire gage-board equipment. The remaining manometers and Bourdon tube gages, etc., were therefore frequently checked for accuracy against this manometer.

The greatest discrepancies in these manometers seemed to result from inaccuracies in the zero settings of the manometer scales. Errors due to this cause might possibly amount to 1½ per cent of the manometer reading for values below 5 lbs./in.^{2} For the higher values of the manometer readings, however, the error as a general rule ranged from 0.1 to 0.7 per cent.

Besides the Bourdon tube gages and the pressure manometers there was installed on the gage board as mentioned above a series of vertical and inclined differential manometers for measuring the flow-meter differentials. For the higher values of the flow-meter differential the vertical manometers were used, but the line was arranged so that for the lower values the inclined differential manometers could be cut in at will. The inclined manometer for measuring the differentials on the acetylene flow meter was mounted so that it could be set at either of two positions. (See Fig. 7.) These differential manometers were mounted with special calibration charts giving the flow-meter differential pressures and a series of curves enabling the determination by rough approximation of the rates of flow of the gas when the differential and the static pressure were known. The charts in question were based upon pressures above atmosphere standardized to 757 mm of mercury. The charts, however, are not fully described, because the readings obtained were only approximate since they did not contain corrections for temperature drop or changes in barometric pressure, nor did they take into account the question of the changing density of such gases as tanked acetylene. The general construction of these charts, however, is clearly indicated in Figs. 7 and 8.

In order to maintain the correct initial zeros for the flow-meter differential manometers, they had connected in their line at their base a small reservoir (*w'*, Fig. 2). This reservoir was so mounted that by means of the lever and set screw the zero of the manometer could be readily adjusted.

It was found necessary to protect these instruments against accidental blowouts due to sudden back pressures caused by unforeseen gas explosions in the gas lines leading to the blowpipe. This was done by a baffle (*w*, Fig. 2) installed immediately over the top of each differential manometer, which interrupted any liquid accidentally blown out and, upon the release of pressure allowed the liquid to flow back into the manometer tube, permitting immediate continuance of the test without serious interference.

In order to complete the records in the test data, the gage board was also equipped with a standard calibrated thermometer, a psychrometer, a barometer, and a stop watch. (See Fig. 7.)

3. THE WELDING TABLE

All welding during the tests was performed upon the welding table illustrated in Fig. 9. This was a wooden frame table approximately 3 feet square, the top of which was composed of fire bricks. On top of the fire bricks was placed a heavy casting channeled for a width of about 6 inches throughout its length. This formed the base upon which all the plates for welding rested during the welding operation. The plates were aligned centrally along this base with the idea that the casting with its grooved surface would permit of better heat radiation along the line of the weld.

As indicated in Fig. 9, the line of the weld was placed directly in front of the welder and the welding was performed from the back toward the operator, thus giving him a full view of the work as it progressed. The welded plates were cut so that the welds were 1 foot in length. Where 2 feet of weld were made continuously, pairs of plates were set in front of each other with a slight space between the individual pairs and with proper allowance for expansion so that the process could be carried from one plate to the other without any interruption. The groove along the base plate facilitated the preheating of the second pair of plates, so that the start upon the second weld was made under conditions practically identical with those which existed when the first pair of plates was finished, a condition equivalent to welding one

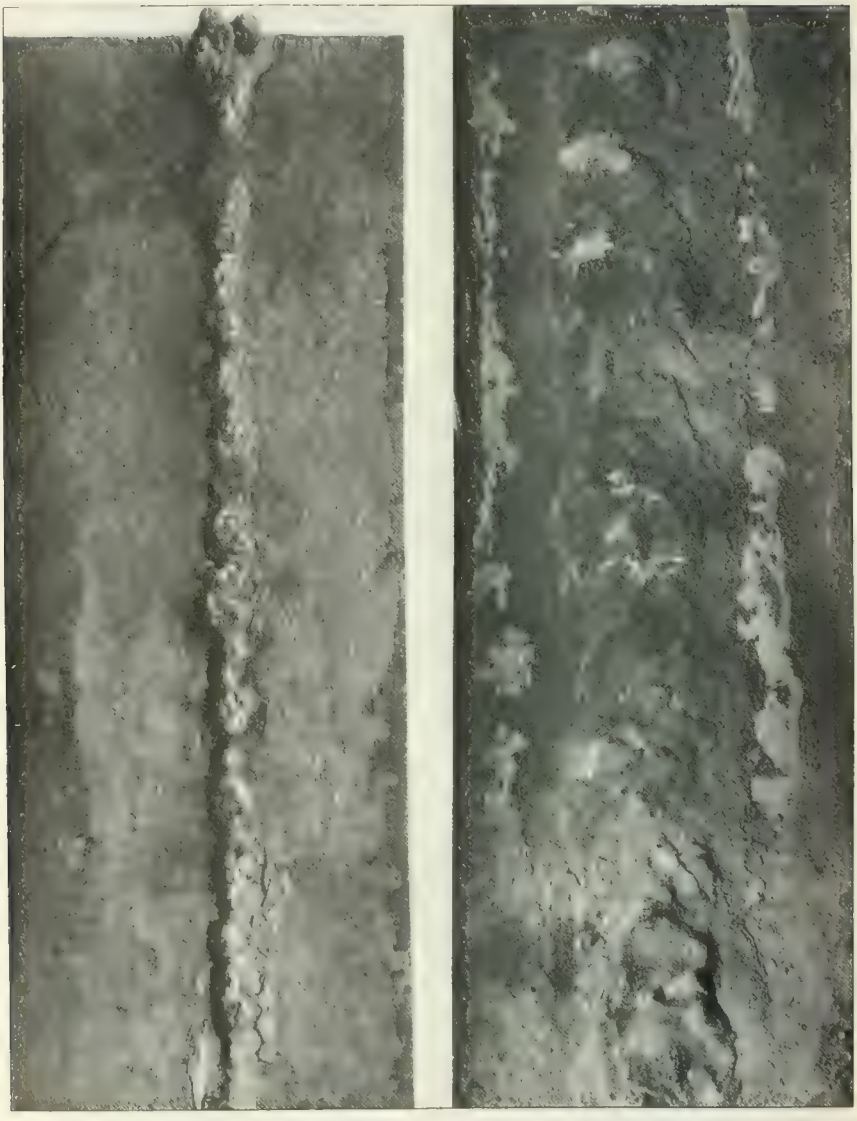


FIG. 10.—*The top and bottom view of an average weld*

2-foot length instead of two 1-foot lengths. Particular care was taken throughout the tests to level the plates before welding and to set them with a proper expansion allowance so that they would come together properly at the completion of the weld without overlapping. Fig. 10 shows the top and bottom view of an average weld. It will be noted that special attention was given to securing good penetration. The micrographs given later show further examples of the good penetration secured (Figs. 65 to 71).

4. THE CUTTING TABLE

In order to minimize as far as possible the personal equation entering into cutting tests all tests were made by a mechanically controlled cutting device installed upon the cutting table. (Fig. 11.) This was a wooden frame table approximately 4 by 12 feet in size with metal bound edges and a fire-brick top. At each end of the table and securely fastened to it were 6 by 6 inch sticks placed crosswise, along the top of which were fastened 1 by 2 inch bars of metal. These latter acted as runners to carry a channel placed with flanges down as illustrated in the figures. The channel held the track upon which the mechanically controlled cutting device operated. Longitudinal motion of the blowpipe could therefore be secured by causing the machine to move along the track and lateral shifts were made by sliding the channel along the runner bars screwed to the 6 by 6 inch end blocks. In order to facilitate quick lateral movement of the channel and insure, where such was desired, that pieces of definite width could be cut, the runner bars had a series of holes spaced conveniently at 2 inches on centers and a set of metal pins which fitted these holes. (See Fig. 1.) By inserting the pins in the proper holes the channel could be instantly shoved over a definite required distance.

This compound arrangement facilitated greatly the making of continuous cuts of 10 feet or upward. The cutting machine being fitted with a reverse motion gear could be operated in a forward direction for the full length of the table, reversed in direction of travel, and at the same time the channel slid over a definite distance and thus the blowpipe made to travel backward for the full length of the table for a cut on a new and adjacent line. (See Fig. 11.)

In order to furnish room for the disposition of the slag formed during the cutting operation, the metal to be cut was supported on

a system of teeth as shown in Fig. 11. These teeth proved extremely satisfactory and made a very convenient method of maintaining the metal in proper position for cutting. They could be shimmed to compensate for warped plates, and due to their tooth construction allowed the passage of slag without in any way blocking or interfering with the cutting operation during the test.

The mechanical guiding and controlling device proved extremely satisfactory and reliable, giving no trouble throughout the entire series of tests. The machine, however, was not arranged to allow of continuous variation of speed, which was necessary for an accurate determination of the maximum cutting speed. For this purpose a combination of series and shunt resistances was attached which enabled the speed to be varied continuously from about 1.5 to 25 feet per minute. For the purposes of the tests the blowpipe holder furnished with the machine was replaced by a special holder that would allow greater latitude in adjustment and would at the same time permit the machine to be adapted to hand-cutting blowpipes of all the various makes.

It was found in preliminary tests that the speed of the machine under all conditions remained practically constant for any particular adjustment. In order to make sure of this, however, the time was carefully noted for the various lengths of cut made during all tests. During the operation of the machine in the entire series of tests there was never noted a variation in speed that justified a rerunning of a test. (See Table 1 for a typical speed record.)

TABLE 1.—Time Speed Record for Cutting ½-Inch Metal, Torch No. 14

Length of cut in feet	Length of time to make cut	Time to cut each 5-foot increment
	Minutes	Minutes
0	0.0	0.0
5	2.73	2.73
10	5.42	2.69
15	8.28	[a] 2.86
20	11.01	2.73
25	13.78	[a] 2.77
30	16.52	2.74
35	19.28	[a] 2.76
40	22.03	2.75
45	24.88	[a] 2.85
50	27.58	2.70

[a] It was found during the tests that from 2 to 4 seconds were consumed in reversing direction of cut at end of each 10 feet of length.

FIG. 11.—*The cutting equipment*

FIG. 1.—*The flash back apparatus*

For convenience in handling, the material used during the cutting tests was cut into lengths of 3 or 5 feet. This necessitated the placing of two lengths of plates in order to secure the lengths of cuts desired in making the tests.

To minimize the delay in passing from the end of one plate to the beginning of another during the cutting operation, a method of preheating, as indicated in Fig. 1, was devised. As the blowpipe approached the end of the cut on one plate a welding blowpipe was ignited and played upon the edge of the second plate in line with the cut, thus preheating its edge so that the cutting blowpipe had a warm plate to attack in passing from the first to the second. It was found that by this method of preheating, bringing just a small spot at the edge of the second plate to a cherry red, the blowpipe could be operated continuously for the full length of the cut, cutting through the junction of billets 6 inches in thickness, without changing or slowing the speed of operation. Where the cuts were of such length as to require several traverses back and forth through the length of the table, such as the 50-foot cut for ½-inch plate, the same preheating with the welding blowpipe was carried out on the ends of the plate. (See Fig. 11.) This preheating therefore permitted the cutting operation to proceed as one continuous operation without any delay or interference from the fact that the test cuts were made on relatively short lengths of plate.

The entire cutting table equipment became practically a mechanically self-operating mechanism, inasmuch as after the maximum speed was obtained the only adjustment required of the operator was that necessary to keep the preheating flames of the cutting blowpipe at a proper distance from the face of the plate. This operation itself would have been unnecessary except for the fact that the cutting operation invariably caused considerable warping of the plate as the cutting proceeded, especially with the thinner plates.

It was the fixed procedure in making the cutting tests to adjust the blowpipe to neutral flame with the cutting oxygen valve at full opening and with the gases delivered at the required pressures. After such adjustment was satisfactorily made one or more preliminary trial cuts were made to determine the maximum speed at which the blowpipe would cut. The trial cuts for the major tests were always made upon a cold plate so that the effect of the heating of the plate in the cutting operation was

made negligible. By means of the mechanical gear shifts of the cutting machine and the resistances mentioned above, the maximum speed at which the blowpipe could operate was readily determined, and having determined this maximum speed the actual cutting operation required for the test was conducted. By the use of the mechanical driving mechanism the heating of the material during a test operation became ineffective inasmuch as the maximum speed was determined upon a cold plate, and from then on to the completion of the test the speed of operation of the cutting blowpipe was maintained constant by the mechanically operated mechanism.

5. THE FLASH-BACK AND SAFETY APPARATUS

(*a*) FLASH-BACK TANKS

The testing equipment also included in the gas lines two commercial flash-back tanks (*f*, Fig. 2, and Fig. 1). These tanks were essentially hydraulically controlled valves which were intended to prevent the propagation of an explosion in the blowpipe or gas line backward toward the gas supply. While it was realized that the installation of the water seal of these flash-back tanks might be considered detrimental, due to the absorption of moisture by the gas, it became evident that their installation was nevertheless a prime necessity as a means of protecting the rather expensive gage-board equipment. It was believed, further, that, inasmuch as the oxygen in use generally came from cylinders that contained more or less water, the passing of the gas through the hydraulic seal of the flash-back tanks would in reality tend to standardize the moisture content in the gas and therefore produce similar effects for all blowpipes.

These flash-back tanks proved extremely satisfactory for the purpose intended in that in several explosions they prevented the propagation of the flame beyond the flash tank. They generally ruptured by the blowing off of the head of the tank during the explosion. As furnished the heads were of rather thick sheet metal, fastened on with bolts. This construction proved to be somewhat dangerous to the operators making the tests, and the tanks were therefore modified in their construction by having a rubber packing and a thin sheet of metal fastened to the top with a heavy annulus. By this construction it was expected that if an explosion developed within the flash tank the thin metal sheet would rupture by tearing and thus minimize danger from flying parts.

(*b*) FLASH-BACK APPARATUS

The tests for freedom from flash back and safety in operation were conducted with the so-called flash-back apparatus. For this series of tests a pair of flash tanks was connected into the hose lines immediately back of the blowpipe. Between the flash tanks and the blowpipe proper two observation boxes were installed. These observation boxes consisted of a glass observation tube inserted in the gas lines and protected by a glass-covered box. The box was of wooden side and back construction with a vented back and a duplex glass face. The interior of the observation box was painted dead black to facilitate observation of the flame propagation. (See Fig. 12.) This equipment was used throughout the entire series of flash-back tests and for the severe flash-back test mentioned later. One further piece of equipment was a vertical sliding blowpipe holder constructed and operated as shown in the figure.

III. MATERIALS USED IN TESTS

1. WELDING ROD

The welding rod used throughout the entire series of tests was secured from the Naval Gun Factory, Washington, D. C. This rod was purchased in July, 1917, under Navy Department Specification 22–W–4. A number of chemical analyses were made to determine the composition, with the following results:

	Per cent
Carbon	0.024 to 0.03
Manganese	.05 to .08
Phosphorus	.01 to .015
Sulfur	.023 to .024
Silicon	.002 to .004
Chromium	Trace
Nickel	Not detected qualitatively
Vanadium	Do.

2. STEEL PLATES FOR WELDING AND CUTTING

The steel plates used for welding were ½ inch and ¾ inch in thickness. The material used for cutting was ½, 2, 6, and 10 inches in thickness. All the material used in both welding and cutting, except the 10-inch material, was furnished through the Engineer Corps, War Department, and was selected with special reference to uniform quality for any particular thickness. The ½-inch material was furnished in plates 3 by 5 feet in size and was used for both welding and cutting tests. For the welding tests

the plates were cut in sections 9 by 12 inches in size. The middle section of each plate was retained as a sample for determining the qualities of the plate. The remaining pieces were used for making welds. During the welding tests it was the practice to use plates that were adjacent to each other in the main or full plate before it was cut into weld specimens, so that as nearly as possible the material used for any particular test would be identical.

The ¾-inch material for welds was received in plates 12 inches wide by 6 feet in length. These plates were cut up into sections 9 inches in length, and for the full width of the plate, that is 12 inches. All specimens for welding tests were finished with a butt joint of the single V 90° included angle type.

For the cutting tests the 3 by 5 feet by ½ inch plates were cut into strips approximately 1½ to 2 inches in width, as indicated in the description of the cutting apparatus above. The 2-inch material for cutting was furnished in sections 2 inches by 6 inches by 20 feet. These were cut, for convenience in handling, into 5-foot lengths, and in test operations were cut lengthwise into sections of 2-inch width. The 6-inch material was shell billet steel furnished in 3-foot lengths and was cut lengthwise in test operations.

Analyses of these materials indicated that they contained approximately the following:

½-inch mild steel plate for welding and cutting tests:	Per cent
Carbon	0.14
Manganese	.32 to 0.36
Phosphorus	.012 to .013
Sulphur	.033 to .055
Silicon	.006 to .012
¾-inch plate for welding tests:	**Per cent**
Carbon	0.25 to 0.27
Manganese	.41 to .48
Phosphorus	.011 to .013
Sulphur	.041
Silicon	.004
2-inch mild steel for cutting tests:	**Per cent**
Carbon	0.19 to 0.20
Manganese	.42 to .44
Phosphorus	.025 to .027
Sulphur	.055
Silicon	.009 to .017
6-inch steel for cutting tests:	**Per cent**
Carbon	0.46 to 0.51
Manganese	.60
Phosphorus	.029 to .038
Sulphur	.064 to .067
Silicon	.13 to .14

For extra heavy cutting it was desired to determine what the various blowpipes could do on special material approximating armor plate in quality. For this purpose the Naval Gun Factory supplied a cast steel billet 10 by 36 by 50 inches in dimensions. This proved to be of the following chemical analysis:

	Per cent
Carbon	0.38
Manganese	1.21 to 1.27
Phosphorus	.038 to .043
Sulphur	.019 to .027
Silicon	.25 to .28
Copper	.96
Nickel	2.62 to 2.70
Chromium	less than 0.01

3. GASES

(*a*) OXYGEN

At the initiation of this investigation it was the intention of the Engineer Corps to supply the gas required for making the tests. The oxygen available at that time had been made by the liquid-air process. Later on the question came up as to the desirability of using gas of some other manufacture because the total amount of gas which would be required for the tests could not be secured from the Engineer's Depot at the Washington Barracks. After due consideration it was decided that it would be advisable to make the tests with oxygen made by the liquid-air process, as it is generally said that this gas as a rule did not run as pure as the electrolytic material and, further, that the impurity was an inert element in contradistinction to the impurity of the electrolytic gas. The decision was based upon the knowledge that the purity of the oxygen had considerable effect upon the cutting efficiency although its effect upon the welding tests is thought to be of minor importance. It was felt that if a blowpipe operated successfully on liquid-air oxygen of lower general average purity, it would operate at least equally as well on electrolytic oxygen of greater purity.

During the preliminary calibrations of the test equipment a number of tanks of oxygen were analyzed, and it was found that the percentage of purity ranged from 97.2 to 99.3, with the average analysis approaching very close to 98.3. Recognizing the effect the purity of the oxygen had upon the cutting efficiency of the blowpipe, it was necessary to establish and maintain as closely as possible a standard oxygen purity for all cutting tests. The average analysis of some 40 tanks of oxygen was therefore

assumed as the standard for this series of tests, this standard being 98.3 per cent pure oxygen.

In order to maintain the standard throughout the series of tests it was the practice to analyze each individual tank of oxygen and to determine its tank pressure. From these data the cylinders were selected so as to furnish as near as possible an average analysis agreeing with the desired standard, that is, 98.3 per cent purity.

After a group of tanks had been thus selected and banked and the manifold tested for leaks, a sample of the mixed gas from the manifold was analyzed for oxygen purity. This latter analysis is the one that is recorded on the log sheets of the various tests as the purity of the oxygen. Throughout the entire series of tests it was found possible to secure the standard required oxygen purity within ± 0.2 per cent.

The apparatus used for the analyses of oxygen was a modified Orsat gas analysis apparatus. The sample was measured over mercury and absorbed by an alkaline pyrogallol solution made up according to directions given by Prof. R. P. Anderson.[2]

The accuracy of analyses according to this method depends somewhat upon the size of the sample taken. When approximately 100 cm^3 of oxygen was taken it was found to be ± 0.1 per cent. This was the size of the sample used in analyzing the oxygen taken for analysis for the entire bank of cylinders. For the analysis of the individual cylinders a smaller sample, that is 25 cm^3, was taken from each cylinder. The accuracy of the method for the smaller sample is considered to be ± 0.2 per cent.

(*b*) ACETYLENE

In initiating these tests it was decided to use tanked acetylene in preference to generator acetylene, as it was known that the greater part of the War Department's field equipment would be operated from cylinders, and it was therefore felt that the comparison of the blowpipes should be made upon the basis of the type of gas with which they would be used in field service.

A commercial acetylene was suggested by the War Department and was therefore adopted as the standard and used throughout the entire investigation. The manufacturers of this gas claim a gas free from sulphur, phosphorus, lime, and water vapor. On account of the injurious effects of sulphur and phosphorus upon

[2] R. P. Anderson, "Reagents for use in gas analysis," Journal of Industrial and Engineering Chemistry, 7, pp. 587–596, July, 1915.

the welding process, frequent tests were made to indicate the presence of these elements by the use of a silver nitrate solution. The results of this test throughout the entire investigation were always negative.

An attempt was also made to determine the acetone content of the gas as it was drawn off at various rates of flow from tanks under different pressures. As far as it could be determined there were no available data on this subject, but for the purpose of this test it was felt that an exact investigation to determine the acetone content of the tanked acetylene gas under the above-mentioned conditions would be far too extensive and costly.

To secure some information on the acetone content a portable Rayleigh type interferometer was used. This instrument is primarily useful for the analysis of binary mixtures or mixtures that can be made equivalent to binary mixtures by the removal of one or more gases. It was thought that this instrument, in offering a method of comparing the refractivity of two gases or gaseous mixtures, would offer at least a comparative determination of the acetone content of the acetylene gas as it was withdrawn from the cylinders.

The instrument is so designed that light passes through two tubes, one of which contains the gas mixture that is to be analyzed and the other contains the mixture without the constituent about which information is desired. To compensate for differences in the refractivity in the gas in the tubes and to measure such differences, a micrometer screw is arranged to move a thin plate of glass in the light passing through one of the tubes. After calibrating the scale on the screw for definite differences in refractivity it is very simple to make a calibration of the instrunent for the gas that is to be analyzed. In the work in connection with this investigation the refractivity of acetone vapor was taken as 1073 by 10^{-7}, the value determined by Prytz.

The standard gas for the use of this instrument—that is, the one in which no acetone was to be present—was obtained by connecting the instrument to an acetylene cylinder containing gas at approximately 250 pound pressure (in order to minimize as much as possible the probable acetone content), the gas thus obtained being passed through a solution of sodium bisulfide to remove such acetone as was present, from which it passed through a drying tube filled with calcium chloride. From this latter tube the pure acetylene gas entered one of the interferometer tubes. The gas to be analyzed for its acetone vapor content was also

drawn from the cylinders, but was passed only through a tube of calcium chloride to free it from water vapor, from which tube it passed to the second tube of the interferometer.

The information gathered by means of this apparatus and given below can not be considered reliable, as it will be affected by impurities in the acetylene or acetone, and, further, by the fact that the refractivity for acetone has not been definitely established, the figures of the different investigators being in disagreement by as much as 2 per cent.

One series of analyses was made upon gas taken from a single cylinder at 25 to 30 pounds pressure to find the effect of the rate of withdrawal of the gas upon the acetone content. It was later found that the flow-meter data indicating the rates of flow for this series of tests were incorrect, so that the exact rates can not be definitely established. For that reason the figures for this series of analyses are not given. The series, however, did as a whole show that the percentage of acetone vapor present increased with the rate of withdrawal of the gas.

A second series of analyses was made on gas taken from the same cylinder of acetylene but at different periods of its discharge with the idea of determining the effect the total pressure had upon the acetone content. In the figures given in the table below for this series of analyses the "theoretical per cent" represents the per cent of acetone vapor that should be in gas confined over liquid acetone under the same pressure at 20° C.

Pressure in pounds	Analyses	"Theoretical per cent"
	Per cent	
250	−1.5	1.3
190	+1.0	1.7
110	1.6	2.8
50	3.4	5.4

As is to be expected, there is a steady increase in the acetone content with decreasing cylinder pressure, but in no case is the percentage as high as that which might be theoretically expected. It is interesting to note in this connection that the first value given for this tank shows a negative acetone content. This negative value is without doubt due to the solution in the liquid acetone of some less refractive gas which more than counterbalanced the refractive effect of the small amount of acetone present. It is quite possible that this suggested phenomenon

is also responsible for the analysis value being consistently lower than that theoretically expected. In this connection there was also the possibility that the smaller amount of acetone vapor present is also due to the condensation effect mentioned under the discussion of density of acetylene gas.

While it is realized that the results mentioned above can not be considered reliable, it is believed they indicate that, with proper care and manipulation, this method of analysis might be used to secure accurate data concerning the purity of acetylene gas withdrawn from compressed-gas cylinders.

IV. SECONDARY STUDIES FOR SECURING DATA FOR INTERPRETING TESTS

1. CORRECTION OF FLOW-METER READINGS FOR TEMPERATURE CHANGES

The early calibration and experiments with the flow meters indicated that for the very low rates of flow the temperature effects would not be sufficient to warrant the use of a thermocouple on the acetylene line. When, however, the gage board was set in its permanent position and exposed to the heat given off during cutting, it was discovered that certain data would be considerably in error unless thermocouples were installed for both lines. All later and by far the greater portion of the tests were made with the thermocouples in both meters. In order that the few earlier tests which were made without the thermocouple in the acetylene meter might be made as fully as possible comparable to the later tests, a careful study was made to determine the average relationship between room temperature and that of the gas passing through the meters. It was learned that a straight-line relation would probably be accurate within at least 1 per cent, and the data secured from this study were therefore used in correcting the gage-board data secured without the use of the thermocouple in the acetylene meter. Later several check runs were made on the few blowpipes to which these corrections were applied, and the legitimacy of the method of the correction fully established.

2. CORRECTION OF BLOWPIPE DATA FOR ERRORS IN BOURDON TUBE GAGE

Once through accident the small manometer was blown out. In order to continue the tests without awaiting the installation of a new manometer, a Bourdon tube gage of from 0–50 lbs./ in.^{2}

range was installed to replace the injured manometer. This gage was put in without having been calibrated, and the tests on blowpipes of three different manufacturers were run with this gage before the compound manometer was repaired, calibrated, and reinstalled. A careful calibration of the Bourdon tube gage temporarily used showed that it was somewhat in error. Calibration corrections were therefore applied to the tests run with this gage. The welding pressures specified for one of these three blowpipes were entirely too high for the operation of the blowpipe, and the pressures used during the tests therefore bore no relation to the manufacturer's specified pressures. The data secured on that blowpipe were therefore not corrected for the errors in the Bourdon tube gage. Smaller calibration correction values applied to the pressures maintained for the other two makes of apparatus indicated that the pressures actually delivered to the blowpipes during the tests in which the Bourdon tube gage was used were practically identical with the specified pressures, the differences being very slight. It was felt, however, that in justice to the manufacturers the data secured should be as accurate as possible, and therefore an extra check weld of 2-foot length in ½-inch material was made for these blowpipes after the reinstallation of the repaired compound manometer.

3. GAS DENSITIES

(a) OXYGEN

Due to the use, in determining the volume consumed in any particular blowpipe test, of the method of substitution for tank weight loss, the question of gas density became of prime importance in the interpretation of test data. The flow-meter observations were also dependent upon a correct knowledge of the densities of the gas in use.

For the oxygen the value obtained by Morley of 1.4290 grams per liter was considered as the most satisfactory. This value was corrected for the average degree of purity of oxygen used throughout the tests, approximately 98.3 per cent, with nitrogen considered as the inert element present. Upon this basis the density of the oxygen was taken as 0.0890 lb./ft.^{3} This value was probably very slightly in error due to the fact that most of the tanks containing oxygen made by the liquid-air process have more or less water in them, which permitted the oxygen to draw over with it a certain amount of water vapor. However, the partial pressure of this water vapor is relatively small for high

tank pressures, and it is not until the tank is almost exhausted that an appreciable effect upon the density of the gas will be noticed. It was therefore not considered necessary to correct the value for the density assumed above for water vapor content, and it is believed that the error involved in this procedure was never greater than 0.5 per cent for all tank pressures above 300 lbs./in.^{2} The tank pressures were seldom lowered below this value during the test operations and in no case below 200 lbs./in.^{2} At the latter pressure the error in the density value due to the neglect of the effect of water vapor is about 0.7 per cent. It is probable then that the error in the interpretations of the test data on the basis of an assumed oxygen density of 0.0890 will in the great majority of the tests be below 0.5 per cent.

(*b*) ACETYLENE

The question of the density to be used for the tanked acetylene was far more complicated. An extended search indicated that practically no information was available concerning the density of acetylene drawn from dissolved acetylene cylinders. It is generally known that the density of the gas is affected by the temperature of the tank, the internal pressures, and the rate of withdrawal of the gas. Inasmuch as it had been decided to use tanked acetylene for this investigation, it was felt that some definite knowledge on this matter was necessary. Accordingly a series of tests was made to determine the probable densities of the gas at various tank pressures. The apparatus used in this determination consisted of the dry gas meter used for calibrating the flow meters with the acetylene balance system used during the tests. The gas was withdrawn from the bank at different rates of flow and was passed through the flow meter and the gas meter to the air. It was passed through the flow meter for a sufficient length of time to insure that the loss in weight of the tanks, as determined by the balance system, would have errors well within the generally accepted range of errors used throughout the investigation. During these runs a thermometer was kept at the outlet of the gas meter and observations of temperature and atmospheric pressure recorded in order that the dry meter volumes could be properly reduced to standard conditions.

Quite a number of runs were made to secure data for determining the density of acetylene. The gas as a rule was drawn from a bank of three tanks, and the principal study was made when it was withdrawn at the rate of approximately 30 cubic

feet per hour, which was considered to be about the average rate of flow actually used during the blowpipe tests. Additional test data were secured for gas withdrawn at the rate of 65 cubic feet per hour and at 11 cubic feet per hour. A short study was further made on the withdrawal of gas from a bank of two tanks at the rate of about 30 cubic feet per hour, which was equivalent to 45 cubic feet per hour for a three-tank bank.

In addition to the study made for this investigation, the Commercial Acetylene Co., through A. H. Ahldin, sent for comparison some additional information from density tests made by them with the Regnault apparatus. This latter work was based upon gas withdrawn from a single cylinder with discharges at the rates of 37.5, 15.7, and 9.3 cubic feet per hour. The values obtained from the latter study have been transformed to agree with the three-tank bank density determinations made for this investigation, giving density determinations for the rates of discharge of 112, 47, and 28 cubic feet per hour, respectively.

The entire data on density determinations thus secured are plotted in Fig. 13, and it will be noticed that the points representing the various rates of flow are designated also with a series of letters indicating the various gas banks from which the data were secured. This latter distinction seemed to be necessary, as the density studies indicated that for similar rates of flow different gas banks gave unlike densities.

A study of the plotted data in Fig. 13 indicates that the density is greatly affected by several conditions. First, it is apparent that it varies decidedly with the tank pressures, increasing rather rapidly as the tank pressure falls. Second, that the density varies with the rate of the withdrawal of the gas and seems to be highest for the lower discharges. Third, a marked difference in densities is especially noticeable at low tank pressures between different banks with the gas discharged at similar rates of flow.

The simplest assumption as to the cause for this variation in density is that the acetylene gas withdrawn from the tank carries with it a varying amount of acetone vapor, the amount of acetone present being that which might be expected in the same volume of acetylene in a system where acetone vapor and liquid acetone are in equilibrium. This assumption, however, does not pretend to account for the full amount of the acetone actually passing from the tank, as the varying densities would indicate that a greater proportion of acetone than that based upon the above assumption passes out with the acetylene. It is probable, there-

fore, that a certain amount of the acetone vapor is carried out mechanically as a fine spray in suspension in the acetylene gas.

It will be noticed in Fig. 13 that quite a number of the density determinations where the gas was withdrawn at 30 cubic feet per hour lie close to a definite curve from point *1C* to point *1B*. If these points are used for a simultaneous solution of acetylene

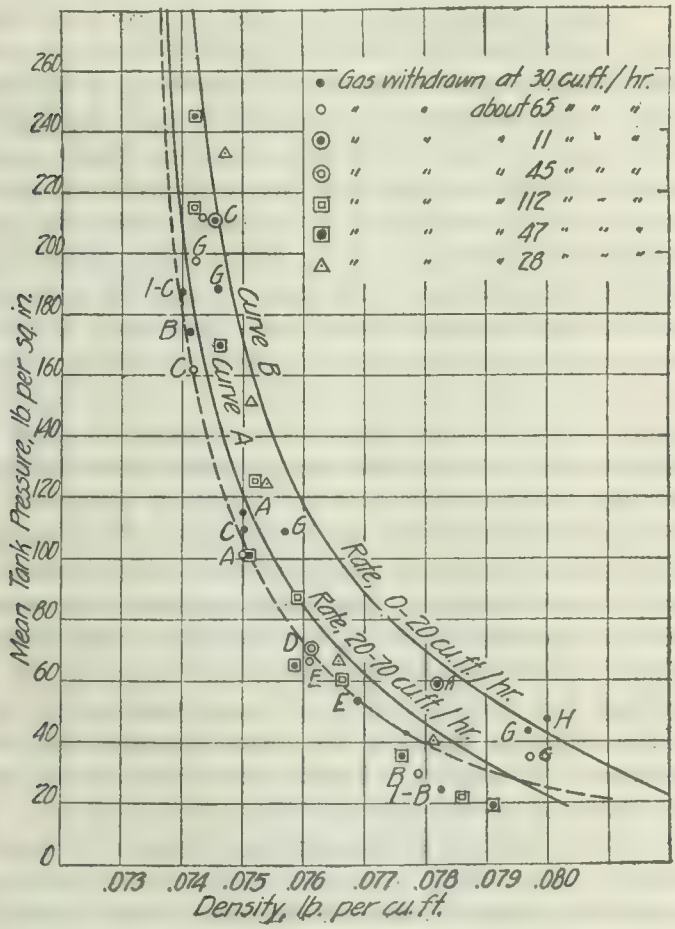


FIG. 13.—*Average density of acetylene*

density and the acetone vapor density, on the basis that both gases are in equilibrium, a value of 0.0725 lb./ft.^{3} is obtained for the acetylene density for the mean simultaneous solution of all of the points except the point *1B*. On the basis of this value and the average value obtained for the acetone density a theoretical density curve indicated by the dotted line in Fig. 13 can be drawn. An examination of the data plotted on the curve

indicates clearly that this assumption is fairly accurate for all tank pressures except the very low values.

As to the effect of the rate of flow upon the density of the acetylene, it seems evident that the lower acetone content for the higher rates of flow is to be most readily explained by the lower temperature of the gas withdrawn from the cylinders when the rate of flow is high. The effect of the lower temperature is to lower the vapor pressure of the acetone and probably collect a portion of the acetone spray within the tanks before it has entirely passed out of the tank. From the data gathered in this investigation it is evident that there is a decided increase of density with the lower rates of flow. This condition has been somewhat corroborated by the data supplied by the Commercial Acetylene Co. The study of the density of the gas withdrawn from tanks at low pressure also indicates that the previous history of the bank affects the values obtained. The bank G and the bank H were used only for welding and welding gas ratio tests, all of which were run upon the upper pressure ranges of the tanks. The rates of flow for these tests were quite high. It will be noticed that considerably more acetone seems to be present for these banks on the lower tank pressures than was present in other banks where low and high rates of flow of equal duration alternated. The density values determined for the higher rates of flow for bank G seemed to indicate however that the bank itself behaved quite differently from the others. The values obtained here seemed to to be in close agreement at the higher rates of flow with the density determinations furnished by the Commercial Acetylene Co. Notwithstanding this, however, it would seem that acetylene withdrawn from different banks at the lower pressures may vary in acetone content by reason of the fact that more or less acetone has been taken from the upper portion of the tank filler in the earlier or initial discharge of the gas from the bank. It is probable also that the unavoidable variations in the density of the filler may retard the escape of the acetone from certain tanks. This latter seems to be borne out by a statement of the Commercial Acetylene Co. to the effect that their experiments seem to indicate that different constructions of the porous fillings have a decided influence upon the density of the gas withdrawn.

In view of the greater discrepancies in the density values at the lower tank pressures it is quite evident that one continuous curve representing the probable average density of the acetylene

withdrawn from the banks could not be used in computing the volume flows. It was therefore decided that the best results would be obtained from the use of the two curves, *A* and *B*, shown on Fig. 13, curve *A* being used for the determination of the densities at the higher rates of flow and curve *B* being used for densities at the lower rates of flow. It is on the basis of the latter two curves that the volume computations in this report were made.

4. PRESSURE DROP IN GAS SUPPLY LINES

The oxygen gas line (*e*, Fig. 2) used for all welding, gas-ratio, and low-pressure cutting tests was a 28-foot length of ¼-inch flexible hose. For the extra heavy cuts (10 inches and above, test 9) a ⅜-inch hose of 26-foot length was substituted for the above. The acetylene gas supply line was a 30-foot length of ⅜-inch flexible hose. Throughout the entire investigation the flash-back tank (*f*, Fig. 2) was always connected in the acetylene supply line. A similar flash-back tank was used in the oxygen gas supply line only in the welding, welding gas ratio, and flash-back tests.

Due to the fact that the length of flexible hose used to transmit the gas from the regulator to the blowpipes was somewhat in excess of that in general use with such apparatus, and, further, inasmuch as the gas lines were supplied with the flash-back tanks, it was felt that the line drop through the hose and the tanks should be determined so that it would be possible to specify the actual pressure delivered at the blowpipe handle for the corresponding static pressure held at the gage board from line *r*, Fig. 2, as the manufacturer's specified regulator pressure. With the lines in the actual condition in which they were to be used and containing identical fittings and flash backs, observations were made by the use of mercury manometers to determine the line drops. The data secured from these observations were used in computing the results of the tests. Due to the fact that the differential manometers were used in several positions and that several different sized orifices were used in the flow meter, it was necessary to plot a number of curves to represent the particular conditions under which the line drop was desired.

In the interpretation of test data the static pressure held at line *r*, Fig. 2, as the manufacturer's specified regulator pressure, has been designated the "mean line pressure." This value has been corrected for the line drop furnished by the plotted data of the above-mentioned curves, and the corrected value is reported

as the "mean torch pressure," which is the pressure at which the gas was delivered to the blowpipe for the particular test in question.

5. LOSS OF PRESSURE THROUGH MANIFOLD

A study of the pressure drops through both the oxygen and acetylene manifolds was made to determine whether any possible difference existed between the pressure indicated on the gage and the mean pressures within the bank. The rates of flow of the acetylene gas were so low that there was apparently no noticeable loss of line drop on the acetylene bank. On the oxygen bank, however, it was found that when the rate of discharge was in the neighborhood of 3000 cubic feet per hour, there was a possible increase of 3 pounds in the pressure as determined from the gage observation made immediately after closing the line valve from the regulator and immediately before closing such line valve. This indicated that for all practical purposes the line drop within the manifold was negligible for all pressures and rates of flow used during the tests, and that it was only when the rate of flow exceeded 3000 cubic feet per hour, and the tanks were being discharged while under approximately full pressure, that any noticeable pressure drop was obtained. Even under these conditions the pressure drop within the tank valves and manifolds did not exceed approximately 3 pounds.

6. HAND VERSUS MACHINE CUTTING

All cutting done during the investigation on the tests of cutting blowpipes, with the exception of the cut for maximum thickness, was done as described above by means of a mechanically-controlled and guided holder. The decision to use this mechanical appliance was based upon three facts. First, it would tend to eliminate the uncertain personal equation of the operator; second, its performance when properly adjusted would be far more regular than could be expected of hand operation; and, third, it was felt that with such a method of operation the maximum cutting speed at which a blowpipe could be operated could be readily fixed and held for the entire test period.

It seemed desirable, however, that some information should be secured as to what might be expected by hand cutting, and with this in mind several cutting tests were run under like pressure conditions and on similar material with a blowpipe that had given rather satisfactory results as used in the cutting machine. The

data of the following table are given to show the comparative rates of cutting on 5 feet of ½-inch material.

TABLE 2.—Hand Versus Machine Cutting

Method	Pressure		Rate of flow		Rate of cutting	Comparison of hand with machine cutting		
						Ratio of rates of flow		Ratio of rates of cutting, hand-machine
	Oxygen	Acetylene	Oxygen	Acetylene		Oxygen	Acetylene	
	Lbs./in.²	Lbs./in.²	Ft.³/hr.	Ft.³/hr.	Ft./hr.	Per cent	Per cent	Per cent
Machine	20.0	3.0	39.5	8.04	61	104	96	73.8
Hand	20.0	3.0	41.1	7.73	45			

The hand cutting was done by the test operator, who has had a number of years experience with both oxyacetylene and oxy-hydrogen blowpipes, and the results show that with the best speed he was able to develop in hand cutting the machine cutting was some 25 per cent faster for the thickness of plate used during the test. This figure does not, of course, necessarily apply to all thicknesses of metal and to all conditions under which cutting operations are performed. It is probably true, however, that the difference between machine and hand cutting, up to what might be considered the commercial limit of oxyacetylene cutting, probably about 10 inches in thickness, will greatly increase with the increased thickness of metal. This statement, of course, refers to conditions where the machine operation can be used to advantage.

7. EFFECT OF THE PURITY OF OXYGEN ON EFFICIENCY IN CUTTING

Several of the blowpipes submitted for cutting tests failed to make cuts for the greater thicknesses of metal. One or two of these have generally been considered as better grade blowpipes, and it was therefore felt that if possible the reason for such failure should be determined. Discussion with the manufacturers of these blowpipes indicated that in at least one instance the blowpipe was initially designed and proportioned for use with an electrolytic oxygen which was on an average of somewhat higher purity. It is known that the increased purity of the oxygen within certain limits has a decided influence on the efficiency of the cutting operation.[3] It seemed desirable, therefore, that

[3] Alexander E. Tucker, F. I. C., "The influence of impurities in oxygen when used for cutting iron and steel," J. of Soc. Chem. Ind., 30, No. 13, p. 779; July 15, 1911.

exact data based upon the apparatus developed for this investigation should be secured with the idea of establishing the probable differences which might be expected for oxygen of various degrees of purity. Difficulties were encountered, however, in securing a supply of tanked oxygen of the various degrees of purity desired. In view of the probable delay involved in completing the investigation, it was decided that no further work along this line would be undertaken.

8. DETERMINATION OF GAS VOLUMES CONSUMED BY LOSS OF PRESSURE INDICATED BY TANK PRESSURE GAGES

In general welding and cutting operations it is quite a common practice to determine approximately the amount of gas consumed on any particular piece of work by means of the loss of tank pressure indicated by gage readings. For oxygen the values thus obtained have proved fairly satisfactory, but acetylene gas volumes thus determined have never been found reliable. In general practice, therefore, the volume of acetylene used has always been taken as equivalent to the oxygen volume thus determined. This practice, of course, can be justified on theoretical grounds.

It was felt that a little study given to this matter might prove of interest, and an attempt was therefore made to determine a constant for both gases which could be considered the equivalent in cubic feet for a loss of 1 pound tank pressure.

The determination of a general useful constant for acetylene seemed to be impossible, as might be expected. As pointed out in the discussion on the density of acetylene, the conditions under which the gas is withdrawn greatly affect the amount of acetone withdrawn with it. Further, it was found that, at the lower pressures especially, there was a time interval effect which made volume determinations from pressure gage readings quite unreliable. This latter element seems to be due to the difficulty with which low pressure gas seeps through the porous filler. As an example of this it was observed on one occasion that the acetylene bank pressure gage at the completion of one day's tests read 70 lbs./in.^{2} with a room temperature of 85° F. The following morning with a room temperature of 84° F the same test gage read 85 lbs./in.^{2}, the gain apparently being due to the fact that, at the earlier reading the gas pressures in the bank had not reached a state of equilibrium. In fact the test gage will nearly always show, a few minutes after the completion of a test, a reading

of from 1 to 3 pounds higher than that taken immediately at the completion of the test.

With the above in mind it was felt that the best approximate constant could be obtained from actual blowpipe test data using the volume by balance as the true volume. From this latter method a constant of 2.2 ft.^{3}/lb./in.^{2} of bank pressure (three tanks, 225 cubic feet capacity) was derived and used in the reduction of test data for determining the volume of gas used, as indicated by gage pressure observations.

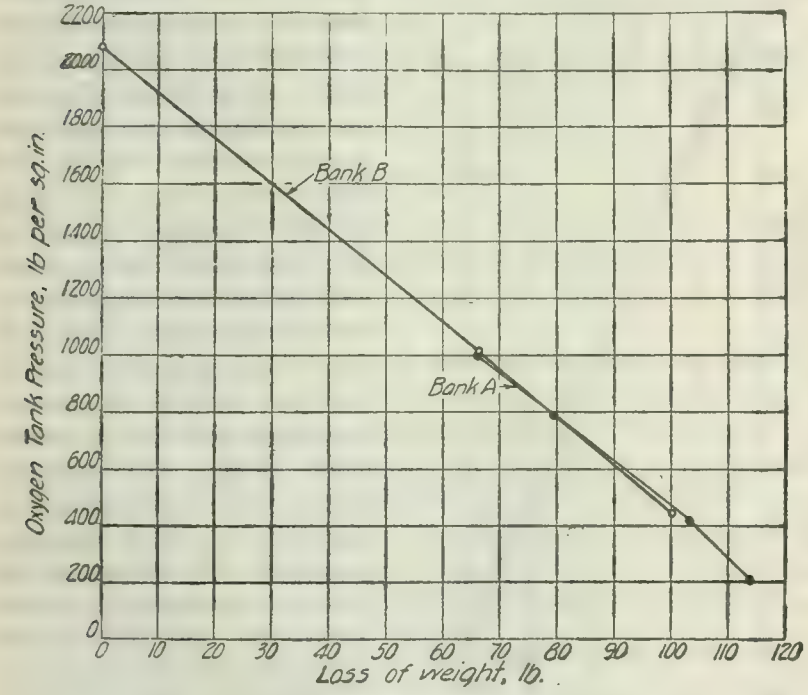


FIG. 14.—*Loss of oxygen corresponding to tank pressure*

In order to determine the oxygen equivalent in cubic feet for a loss of 1 lb./in.^{2} bank pressure, a study was made of two representative oxygen banks. The total loss of weight of the banks is shown plotted against gage pressure in Figs. 14 and 15. Two oxygen banks were used by reason of the fact that it was known that occasionally water may take up considerable space in the bottoms of the oxygen tanks. The two banks were therefore studied in order to determine whether any great differences existed between the banks themselves. A slight difference in the capacity of the banks seemed to be perceptible. For the purpose of approximate volume determinations the selection of a mean value for the

number of cubic feet per pound per square-inch gage pressure seemed warranted. This value was taken as 0.68 ft.^{3}/lb./in.^{2} of bank pressure or gage pressure for a 7-tank oxygen bank with tanks of 200 cubic feet capacity.

V. DESCRIPTION OF THE TESTS

The investigation was started with the idea of submitting each manufacturer's equipment to a much more extensive series of tests. It was found, however, that the proposed series was excessive from the time standpoint. One of the most serious drawbacks as a time consuming element was the fact that a very large percentage of the blowpipes submitted for test would not operate with the pressures specified by the manufacturers. This condition was probably due to the fact that it is quite a customary practice to recommend setting the regulator pressures 3 to 5 pounds higher than the specified blowpipe pressures. By throttling the gases at the blowpipe handle valves the operator insures having sufficient pressure available at all times to maintain the required velocity of exit of the gases at the tip end. He is therefore enabled to compensate for pressure fluctuations due to irregular action of the regulator, thus tending to minimize the development of flash backs. The specifications for the tests distinctly stated that at least one of the blowpipe handle valves must be maintained at full opening during a test. It was only by such a procedure that the gas consumption of a blowpipe could be definitely ascertained. For a great many of the blowpipes the pressures were too high to

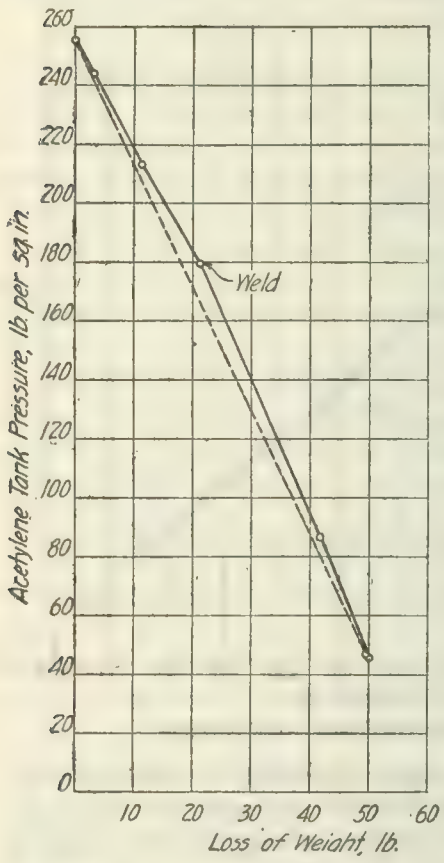


FIG. 15.—*Loss of acetylene corresponding to gage pressure*

enable the maintenance of a stable flame with one of the handle valves at full opening.

Another quite serious source of trouble from the standpoint of time consumption was that due to leakage, necessitating the dismantling and repacking of the valves.

In order, therefore, that the entire investigation might not require an undue length of time it was decided that attention should be devoted only to the so-called primary tests, consisting of the welding, cutting, gas ratio, and flash-back tests. Such proposed tests as the variation of pressure within the blowpipe head, etc., were therefore abandoned. On the basis of the above the following schedule of tests was adopted and all blowpipes tested during this investigation were submitted to it.

1. WELDING TESTS

All blowpipes reported upon were submitted to five welding tests, designated respectively as tests 1–a, 1–b, 1–c, 1–d, and 2. All the tests numbered 1 were made with ½-inch plate. Test 2 was a weld with ¾-inch plate.

Tests 1–a and 1–b were made with the tip sizes and pressures specified by the manufacturer if this were possible. For both of these tests a 2-foot length of weld was made. These tests were identical in all respects, with the exception that an attempt was made to evaluate the personal equation by using different operators. For tests 1–c and 1–d a 12-inch length of weld was made. Both of these welds were made by the operator who made the weld of test 1–a, the idea being to maintain as nearly constant a personal equation for this series of tests as possible. Test 1–c was run with the same size tip as 1–a, but with pressures (both oxygen and acetylene) 50 per cent in excess of the pressures used for test 1–a. Test 1–d was carried out similarly to 1–c except that the pressures were 25 per cent below those used in test 1–a.

As mentioned above, the pressures specified by the manufacturer very often gave an exit velocity to the gas too high to permit of maintaining a stable flame at the blowpipe tip. In such cases the manufacturer's representative was requested to furnish a modified pressure that would enable the maintenance of a stable welding flame. Very often the modified pressure thus determined upon would not permit of the application of test 1–c; that is, a test with a 50 per cent increase in pressure in both gas lines. It was customary, therefore, in such cases to modify the test procedure and incorporate as a test in place of test 1–c, test

1–e, which was run under identical conditions with the above test, with the exception that the pressure on both gas lines was reduced to 50 per cent of the pressure used to make test 1–a.

Tests 1–c, 1–d, and 1–e were incorporated to show the effects of increased or decreased pressures on the operation and economy of the blowpipe. Such excess or decreased pressures are found to be quite common in many welding operations, due to carelessness on the part of the operator in setting regulator pressures or to imperfect regulator action. It was felt that a properly designed blowpipe should be capable of adjustment over a considerable range for any specified tip size. It was hoped in the investigation to secure data that would either verify this assumption or prove that it was absolutely essential to maintain exact pressures for satisfactory blowpipe operation.

Test 2 was a 12-inch length of weld of ¾-inch mild steel plate. This weld was made in all cases by the operator, who made the weld of test 1–b. This test was selected as indicating the probable results to be obtained with a blowpipe in heavy welding, and with test 1 it was felt that it would give a fair idea of the adaptability of the blowpipe for welding purposes. The tips for welding ½ and ¾ inch plate were selected as being the tips used respectively for the average size weld and for the maximum size weld and therefore the best general average for determining the blowpipe's efficiency and safety.

2. CUTTING TESTS

The cutting blowpipes reported upon have been submitted to eight cutting tests, four of which may be considered primary. In each of the cutting tests a length of cut was made sufficient to determine accurately the gas consumption of the blowpipe. Test 6–a consisted of cutting continuously 50 linear feet of ½-inch mild steel, test 7 was a continuous cut of 10 feet of 2-inch mild steel, test 8 was a continuous cut of 6 feet of 6-inch medium steel, test 9 was a cut by hand for maximum thickness.

As indicated above in the description of the cutting test equipment, each test cut was preceded by a preliminary trial cut to determine the maximum speed at which the blowpipe would cut with the tip size and pressures specified by the manufacturer for the particular thickness of metal. It was found during these trial cuts that the same condition existed relative to the cutting blowpipes that was found to exist with the welding blowpipes, that is, that a great many of the pressures specified by the manu-

facturers for cutting particular thicknesses of metal were not suitable for the work intended. One of the main troubles seemed to be that the majority of the blowpipes were operated with preheating flames of too high intensity.

Throughout the entire series of cutting tests all blowpipes were operated similarly, that is, at least one of the gas control valves for the preheating flames was at full opening, the preheating flames were adjusted to neutral with the oxygen cutting valve at full opening, and all cutting operations were started with this latter open. Some criticism of this procedure developed from time to time, but throughout the entire series of tests it was found that at no time were any detrimental results produced, and that even on cuts as heavy as 16 inches no trouble was found in starting with the oxygen cutting valve open. This method was adopted to insure the maximum capacity of the blowpipe throughout the entire test period, and, further, to enable the maintenance of accurate pressures throughout the test and insure accurate data as to the maximum gas consumption during the cutting operation.

Test 9, cutting for maximum thickness, was originally conducted with the idea of submitting the blowpipe to a test for the maximum thickness specified by the manufacturer as permissible for his blowpipe. It was found during the course of the investigation that the great majority of the blowpipes would not successfully cut the so-called maximum thickness specified by the manufacturers. Experience during the investigation seemed to indicate that the probable maximum commercial limit for economic heavy cutting operations when the cut is made in one direction only with an ordinary stock blowpipe, lay in the neighborhood of 10 inches. It was therefore decided that all blowpipes should be submitted to the same maximum or heavy cut. A cut in material 10 inches in thickness, of 12 to 18 inches in length was therefore made as a supplementary heavy cut test for each blowpipe.

The tests for maximum thickness of cut were made by hand. On account of the possibility of damage to the gage-board equipment caused by heavy slag showers from pockets, etc., produced during cutting, this test was conducted out of doors. The original tests for maximum thickness were therefore made on different days with the temperature of the metal varying from 35 to 68° F. After the decision had been made to submit all blowpipes to cutting tests of identical thickness, it was decided to minimize the effects of varying temperature as much as possible on the heavy cut by making all cuts during the course of one or two days. By

this procedure only a slight temperature change occurred. Particular care was also exercised to insure that a blowpipe was not submitted to test on pieces of the steel ingot used for this work that were still warm from a test of another blowpipe. The test for heavy cutting—that is the special 10-inch cut just described—is listed as test 9 in the data that follow.

Besides the primary tests it was thought desirable also to determine the efficiency of the blowpipe as it was affected by variations in the oxygen pressure supplied. For this purpose a series of supplementary tests listed as tests 6–b, 6–c, 6–d, and 6–e were conducted. These were made on ½-inch mild steel plate and a cut of 10 linear feet made for each test. In this entire series of tests the tip sizes and the conditions of cutting were identical with test 6–a listed above, with the exception that the oxygen pressures alone were modified. These were increased 50 and 25 per cent and decreased 25 and 50 per cent respectively for tests 6–b, 6–c, 6–d, and 6–e. No change was made in the acetylene pressure. A slight aspirating effect for the increased oxygen pressure was expected and was shown in the volume of the acetylene used.

3. GAS-RATIO TESTS

One of the prime essentials of a good welding blowpipe is its so-called gas ratio—that is, the ratio of the volume of oxygen to the volume of acetylene consumed. Theoretically a properly adjusted blowpipe requires equal volumes of both gases, giving a one-to-one ratio. In order to establish the ratios of the blowpipes under test, each one reported upon was submitted to a series of gas-ratio tests, numbered 5–a–1 and 5–b.

For these tests the blowpipe was allowed to burn freely in air with the same tip sizes and pressures as were used in the welding tests mentioned above. The blowpipe was supported upon a bracket stand with the tip in horizontal position. All gas-ratio tests were made upon the tips used for welding, and no attempt was made to clean them before making the tests other than to blow them out with a rather high oxygen pressure. These tests might be expected to indicate the gas consumption of the blowpipe with the flame burning undisturbed from accidental obstructions, such as slag adhesions to the ends of the tips, etc., which occur in actual welding practice. The discussion of the welding blowpipe furnishes additional interesting information concerning the gas-ratio tests.

4. FLASH-BACK TESTS

In order to determine the probable safety of operation of a welding blowpipe and, further, to secure information concerning the permanency of construction of the tip and blowpipe head, the blowpipes reported upon were submitted to two types of flash-back tests.

One series of tests, designated 3–a and 3–b, flash-back tests on the tips used for welding ½-inch metal and ¾-inch metal, respectively, consisted of the standard series of tests used by the Underwriters' Laboratories for determining the freedom from flash back and the safety of the welding blowpipe. Each of these tests consisted of four distinct operations. The first three of these operations were carried out as follows: After being properly adjusted to neutral flame the blowpipe was tested for flash back by drawing the tip at varying angles across the surface of and finally pressing the tip end firmly against certain materials. For this test a cold steel plate, a fire brick, and a piece of wood were used. Finally, the tip was used to make a pool of molten metal in a cast-iron block, flux being used to assist in maintaining the fluid condition of the metal and the tip suddenly plunged into the pool of metal.

Another series of tests intended to determine the permanency of construction and designated as the "severe flash-back test," test 4, was carried out by supporting the blowpipe on a vertical sliding carriage in such a manner that at the proper instant the ignited and carefully adjusted blowpipe could be lowered so that the end of the tip was directly over the center of a hole 2 inches in diameter and 2 inches deep, drilled in a heavy cast-iron block (Fig. 12).

Experiments in the development of the severe flash-back test indicated that this procedure, by causing a reflection of the flame up about the head of the welding blowpipe and concentrating the entire flame about the tip and blowpipe head, not only caused the development and propagation of a true flash back with all blowpipes but further caused the tip to become red hot and tended to melt the brazed joint at the juncture of the gas tubes and the blowpipe head. While this test might be considered rather severe, inasmuch as it will not occur in general, ordinary practice, the condition is nevertheless likely to occur to rather a large extent in welding in corners, etc. Further, by being easily duplicated as to conditions it permitted the obtaining of definite information as to the effects of excess heat upon the blowpipe tip and head.

In all the tests for flash back the flash-back observation boxes mentioned above in the description of the testing apparatus were used and careful notes taken to record the behavior of the conditions developed during the tests. In the case of the severe flash-back test, test 4, particular attention was also paid to the effect of excess heat upon the blowpipe heads and as to whether the gases could be shut off and the blowpipe relighted after the flash back had occurred without the necessity of cooling the blowpipe with water.

In all the flash-back tests the blowpipe was operated under the identical conditions at which it was supposed to be used during the welding operations, and after the initial adjustment of the flame no further adjustments were permitted. Following also the procedure in the welding tests, at least one of the blowpipe handle valves controlling the gas had to be at full opening.

In first carrying out the severe flash-back tests the investigation indicated from time to time minor points for consideration which had not been thought of in previous tests. It was therefore decided to rerun the entire series of severe flash-back tests so that each blowpipe would be submitted to identical tests and observed in like manner. The results of the latter tests only are recorded in this report.

5. LOG SHEET RECORDS OF TESTS

It has not been thought advisable, in the light of the discussions following, to incorporate as part of the report the complete set of test log sheets, especially as there are over 1900 of them. In order that an idea may be had of the amount and extent of the data taken for each test, a few typical log sheets representing the data taken for one manufacturer's apparatus are reproduced in Figs. 16 to 31.

VI. DISCUSSION OF TEST RESULTS

1. GENERAL BASIS OF CONSIDERATION

For convenience in analysis the results of these tests have been summarized in a series of tables (Tables 3 to 7) and plotted graphs, Figs. 32 to 41 and 51 to 60. In the plotted summaries are shown the results of tests on blowpipes from 14 different manufacturers, each represented by a designating number. The group submitted to test comprises most of the better-known and more widely advertised makes of apparatus on the American market and includes besides those voluntarily submitted the three makes of apparatus that were withdrawn by their manufacturers because of disagreement over methods of test procedure. Samples of these latter three were purchased for test purposes.

Date 3-17-20
Computed by A.T.D
Checked by M.L.C

Sheet No. 1

Summary of
Welding Test No. 1-G

Style of Torch L No. 7860
Manufactured by

Thickness of Plate 1/2 in. Length of Weld 24 in.
Time to Make Weld 44 min. 3 sec.
Linear feet Welded per Hour 2.72
Characteristics of Welded Material
1. Tensile Strength 32000 Lb. per Sq. In.
2. Ductility. Bent to Included Angle of 43 Degrees
3. Structure of Weld Slightly crystalline adhesion

Pressure, Gas Consumption and Rate of Flow

	Oxygen	Acetylene
Mean Line Pressure. Lb. per Sq. In.	6.25	8.00
Mean Torch Pressure. Lb per Sq. In.	5.32	7.35
Weight of Gas Used. Pounds	3.812	2.620
Volume in Cu. Ft per Hour Determined From		
1. Weight of Gas Used	42.85	35.31
2. Flow Meter Data	42.8	36.3
3. Test Gage Readings	37	37
Mean Rate of Flow by Flow Meter	(58.4)*	(48.1)*
Cubic Feet per Hour	58.2	49.5
Volume Ratio Oxygen to Acetylene		
1. By Weight Density Method	1.21	
2. By Flow Meter	1.18	

* By Balance

FIG. 16.—*Log sheet No. 1, summary—welding*

Date 3-17-20
computed by A.H.S.
checked by L.R.S.

Sheet No. 2

Summary
of
Test No. 19

Record of Torch Consumption

Time Minutes	Rate of Flow Cu. Ft. per Hour Oxygen	Acetylene	Volume Ratio Oxy. : Acet.	Time Minutes	Rate of Flow Cu. Ft. per Hour Oxygen	Acetylene	Volume Ratio Oxy. : Acet.
1	60.2	50.8	1.18				
3	58.9	50.0					
6	58.7	49.9					
9	58.6	49.9					
12	58.5	49.7					
15	58.2	49.4					
18	58.0	49.6					
21	58.0	49.6	1.17				
24	58.0	49.9					
27	57.8	49.3					
30	57.8	48.9					
33	57.6	49.0					
36	57.6	48.8					
39	57.6	48.7					
42	57.4	48.7	1.18				

Remarks.

FIG. 17.—*Log sheet No. 2, summary—welding*

Date 3-17-20 Sheet No. 3
Recorder Waite
Observer

Weighing Table Log Sheet
for
Welding Test No. 1-a

Style of Torch L No. 7860
Manufactured by

Serial No. of Weld 6-a-1.6-a-2 Operator E.A.J
Thickness of plate 1/2 in. Length of Weld 2 Ft. Tip No. 9
Manufacturer's specified pressures:
Oxygen 6.5 lb. per sq. in. Acetylene 8 lb. per sq. in.
Pressure at which gas was delivered to torch line:
Oxygen 6.25 lb. per sq. in. Acetylene 8 lb. per sq. in.
Number of Gas Tanks in bank: Oxygen 7 Acetylene 3
Average analysis of gas in bank: Oxygen 98.3 Acetylene (97.8)

	Weight of Plates	Weight of Welding Rod
Before Welding	58.5 lb.	2005 grams
After Welding	60.5 lb.	980 grams

Gases used as determined by loss in weight of Tanks:
Oxygen 1730.2 grams Acetylene 1189.5 grams
Time to make weld 44 min. 3 sec.
Remarks:

FIG. 18.—*Log sheet No. 3, weighing table summary—welding*

Date 3-17-20
Recorder Waite
Observer

Sheet No. 4

Log Sheet
Test No. 1-a

Style and No of Torch L 7860
Mfg. by

Operator E A I

		Oxygen		Acetylene	
		Scale Reading	Zero	Scale Reading	Zero
Initial		4.55		24.42	
		12.63	8.55	32.52	28.52
		4.40	8.31	24.63	28.50
		11.83	8.05	32.23	28.46
		4.16		24.78	
True Zero			7.79		28.54
Final		11.23		23.86	
		3.10	7.08	31.08	27.49
		10.91	7.10	23.96	27.48
		3.48	7.13	30.92	27.52
		10.65		24.28	
			7.10		27.49
Displ. of Zero			.69		1.05
Sensitivity			1.82		1.42
Value of Displ			12.5		14.9
Loss per minute			.19		.10
Time			44 min.	3 sec.	
Value of Loss			-8.3		-4.4
Total Correction			4.2		10.5

	Oxygen	Acetylene
Replacement Weight	1726.0	1179.0
Correction	4.2	10.5
True Weight	1730.2	1189.5

FIG. 19.—*Log sheet No. 4, weighing table—welding*

Date 3-17-20
Recorder Deming
Observer Deming-Sweetman.

Sheet No. 5

Gage Board
Log Sheet
for
Welding Test No. 1-a

Torch manufactured by

Serial No. of Weld 6-a-1 6-a-2 Operator E.A.I

Thickness of Plate ½ in. Length of Weld 2 ft. Tip No. 9

Manufacturer's specified pressures:

Oxygen 6.5 lb. per sq. in Acetylene 8 lb. per sq. in.

Pressure at which gas was delivered to torch line:

Oxygen 6.25 lb. per sq. in. Acetylene 8 lb. per sq. in.

Tank Pressures:	Oxygen	Acetylene
At start	845	193
At end	790	176

	Temperature (Room)	Barometer	Humidity
At start	70.9	759	.49
At end	72.0	753	.50

Position of Manometers: High — Low — Vertical O.A

Orifice No _____ Oxygen V Acetylene IV

Temperature reading of orifice:	Oxygen	Acetylene
At start	.78 m.v.	.78 m.v.
At end	.78 m.v.	.78 m.v.

Time required to make weld 44 min. 3 sec.

Remarks:

FIG. 20.—*Log sheet No. 5, gage board summary—welding*

Date 3-17-20 Sheet No. 6
Recorder Deming
Observer Deming-Sweetman

Pressure and Flow Meter Log Sheet
for
Welding Test No. 1-a

Torch manufactured by _____

Serial No. of Weld 6-a-1, 6-a-2 Operator E.A.J Tip No. 9

Pressure at which gas was delivered to torch line:

Oxygen 6.25 lb per sq in Acetylene 8 lb. per sq. in.

Time	Poten	Pres at Reg. Oxygen	Pres at Reg. Acety	Flow meter differentials Oxygen	Flow meter differentials Acety.	Time	Poten	Pres at Reg. Oxygen	Pres at Reg. Acety.	Flow Meter differentials Oxygen	Flow Meter differentials Acety.
1	78	6.30	8.06	132.0	75.9						
3		6.25	7.99	126.4	73.6						
6		6.25	8.00	125.6	73.2						
9		6.25	8.00	125.2	73.1						
12		6.25	8.00	124.8	72.8						
15		6.25	8.00	123.2	71.7						
18		6.25	8.01	122.7	72.3						
21		6.25	7.98	122.8	72.3						
24		6.25	8.00	122.4	73.2						
27		6.25	8.02	121.8	71.6						
30		6.25	7.90	121.5	70.5						
33		6.25	8.05	121.4	70.8						
36		6.25	8.00	120.8	70.0						
39		6.25	7.99	120.7	69.8						
42	78	6.25	8.00	120.0	68.7						
Mean		6.25	8.00	123.4	72.0						

FIG. 21.—*Log sheet No. 6, gage board—welding*

Date 3-17-20
Computed by A. H.S.
Checked by H.W.

Sheet No. 7

Computations
Test No. 1-a

		Oxygen	Acetylene
Mean Line Pressure.	Lb. per Sq. In.	6.25	8.00
Line Drop.	Lb. per Sq. In.	.93	.65
Mean Torch Pressure.	Lb. per Sq. In.	5.32	7.35
Mean Atmosphere.	Lb. per Sq. In.	14.62	
Value of Flash Back Head.	Lb. per Sq. In.	.10	.10
Absolute Flow Meter Pressure.	Lb. per Sq. In	20.97	22.72
Mean Potentiometer Reading.	Millivolts	.78	.78
Mean Temperature Factor.		1.000	1.006
Weight of Gas Used.	Pounds	3.812	2.620
Density of Gas.	Lb. per Cu. Ft.	.0890	.0742
Volume of Gas by Balance.	Cubic Feet	42.85	35.31
Mean Rate of Flow by Flow Meter.	Cu. Ft. per Hr.	58.21	49.5
Duration of Test .734 Hours			
Volume of Gas by Flow Meter.	Cubic Feet	42.75	36.3
Loss of Tank Pressure.	Lb. per Sq. In.	5.5	1.7
Cubic Feet per Pound of Tank Pressure.		.68	2.2
Volume of Gas by Test Gage.	Cubic Feet	37	37
Volume Ratio by Balance.		1.21	
Volume Ratio by Flow Meter.		1.18	
Length of Weld ~~or Cut~~ in Feet.		2.0	
Linear Feet per Hour Welded ~~or Cut~~.		2.72	
Mean Tank Pressure.	Lb. per Sq. In.		184.5

FIG. 22.—*Log sheet No. 7, computations—welding test No. 1-a*

Date 3-17-20
Computed by A.H.S.
Checked by T.L.W

Sheet No. 7

Computations
Test No. 1-b

		Oxygen	Acetylene
Mean Line Pressure.	Lb. per Sq. In.	6.25	8.01
Line Drop.	Lb. per Sq. In.	.94	.60
Mean Torch Pressure.	Lb. per Sq. In.	5.31	7.41
Mean Atmosphere.	Lb. per Sq. In.	14.56	
Value of Flash Back Head.	Lb. per Sq. In.	.10	.10
Absolute Flow Meter Pressure.	Lb. per Sq. In.	20.91	22.67
Mean Potentiometer Reading.	Millivolts	.79	.81
Mean Temperature Factor.		.999	1.002
Weight of Gas Used.	Pounds	3.600	2.379
Density of Gas.	Lb. per Cu. Ft.	.0890	.0744
Volume of Gas by Balance.	Cubic Feet	40.45	32.00
Mean Rate of Flow by Flow Meter.	Cu. Ft. per Hr.	58.65	47.7
Duration of Test .686 Hours			
Volume of Gas by Flow Meter.	Cubic Feet	40.21	32.71
Loss of Tank Pressure.	Lb. per Sq. In.	55	14
Cubic Feet per Pound of Tank Pressure.		.68	2.2
Volume of Gas by Test Gage.	Cubic Feet	37	31
Volume Ratio by Balance.		1.26	
Volume Ratio by Flow Meter.		1.23	
Length of Weld ~~or Cut~~ in Feet.		2.0	
Linear Feet per Hour Welded ~~or Cut~~		2.91	
Mean Tank Pressure.	Lb. per Sq. In.		164

FIG. 23.—*Log sheet No. 7, computations—welding test No. 1-b*

Date 5-27-20
Computed by A.T.D.
Checked by M.L.C.

Sheet No. 1

Summary
of
Cutting Test No. 60

Style of Torch C No. 1362
Manufactured by

Thickness of Cut 1/2 in. Length of Cut 20 ft.
Time Required to Make the Cut 13 min. 57 sec.
Linear Feet Cut per Hour 86.0
Type of Cut Secured:
Top Edge Smooth
Face of Cut Smooth
Drag 5/8 inch
Kerf 1/16 inch
Slag Brittle, slightly metallic slag. Clean bottom edge.

Pressure, Gas Consumption and Rate of Flow

	Oxygen	Acetylene
Mean Line Pressure - Lb. per Sq. In.	25.00	3.02
Mean Torch Pressure - Lb. per Sq. In.	24.70	2.98
Weight of Gas Used - Pounds	1.095	0.179
Volume in Cu. Ft. ~~per Hour~~ Determined from		
1. Weight of Gas Used	12.30	2.41
2. Flow Meter Data	12.2	2.4
3. Test Gage Readings	—	—
Mean Rate of Flow by Flow Meter	(52.9)[B]	(10.4)[B]
Cubic Feet per Hour	52.5	10.2

[B] By Balance.

FIG. 24.—*Log sheet No. 1, summary—cutting*

Date 5-27-20
computed by A.H.S.
checked by L.R.S.

Sheet No. 2

Summary
of
Test No. 6A

Record of Torch Consumption

Time Minutes	Rate of Flow Cu. Ft. per hour Oxygen	Acetylene	Volume Ratio Oxy : Acet	Time Minutes	Rate of Flow Cu. Ft per Hour Oxygen	Acetylene	Volume Ratio Oxy : Acet.
1	53.6	11.8					
3	52.1	11.8					
Flame readjusted							
5	52.5	10.7					
7	51.6	11.1					
Flame readjusted							
9	53.4	9.1					
11	51.9	7.8					
13	52.0	8.7					

Remarks. _____

FIG. 25.—*Log sheet No. 2, summary—cutting*

Date 5-27-20
Recorder Waite
Observer

Sheet No: 3

Weighing Table Log Sheet
for
Cutting Test No. 6-a

Style of Torch G No. 1362
Manufactured by

Thickness of material cut ½ in. Length of cut 20 ft Tip No. 1
Manufacturer's specified pressures:
Oxygen 25 lb. per sq. in. Acetylene 3 lb. per sq in.
Pressure at which gas was delivered to torch line.
Oxygen 25 lb per sq in. Acetylene 3.02 lb. per sq in.
Number of Gas Tanks in banks: Oxygen 7 Acetylene 3
Average Analysis of gas in bank: Oxygen 98.34 Acetylene (97.7)
Gases as determined by loss in weight of tanks:
Oxygen 496.4 grams Acetylene 81.1 grams
Time required to make cut 13 min. 57 sec.
Remarks:

FIG. 26.—*Log sheet No. 3, weighing table summary—cutting*

Date 5-27-20
Recorder Waite
Observer

Sheet No. 4

Log Sheet
Test No 6a

Style and No. of Torch C-1362
Mfg by

Operator R. L. W.

	Oxygen Scale Reading	Oxygen Zero	Acetylene Scale Reading	Acetylene Zero
Initial	16.52		37.84	
	8.96	12.66	29.10	33.42
	16.19	12.62	37.64	33.43
	9.14	12.59	29.36	33.46
	15.89		37.50	
True Zero		12.56		33.46
Final	16.60		30.07	
	8.84	12.65	37.79	33.98
	16.33	12.65	30.30	33.97
	9.12	12.67	37.50	33.95
	16.12		30.50	
		12.66		33.97
Displ of Zero		-.10		-.47
Sensitivity		14.70		14.71
Value of Displ		1.47		-6.91
Loss per minute		.073		.00
Time		13'-57"		13'-95"
Value of Loss		-.103		
Total Correction		-1.57		-6.91

	Oxygen	Acetylene
Replacement Weight	498.0	88.0
Correction	-1.6	-6.9
True Weight	496.4	81.1

FIG. 27.—*Log sheet No. 4, weighing table—cutting*

Date 5-27-20 Sheet No. 5
Recorder Deming
Observer Deming-Sweetman

Gage Board
Log Sheet
for
Cutting Test No. 6-a

Torch manufactured by _____ Operator R.I.W.
Thickness of plate cut ½ in. Length of cut 20 ft Tip No. 1
Manufacturer's specified pressure.
Oxygen 2.5 lb. per sq in. Acetylene 3 lb. per sq. in.
Pressure at which gas was delivered to torch line:
Oxygen 2.5 lb. per sq in. Acetylene 3.02 lb. per sq. in
Tank Pressures: Oxygen lb. per sq in. Acetylene lb. per sq in.

At start	1095	262
At end	1080	260

	Temperature (Room)	Barometer	Humidity
At start	72.1	153.0	.89
At end	73.5	152.5	.89

Position of manometers: High ——— Low A Vertical O
Orifice No. Oxygen V Acetylene IV
Temperature reading of orifice:

	Oxygen	Acetylene
At start	.80 m.v.	.82 m.v.
At end	.80 m.v.	.82 m.v.

Time required to make cut 13 min. 57 sec.
Remarks 2.7 high gear. No resistance.

FIG. 28.—*Log sheet No. 5, gage board summary—cutting*

Date 5-27-20 Sheet No. 6
Recorder Deming
Observer Deming-Sweetman.

Pressure and Flow Meter Log Sheet for Cutting Test No. 6-a

Torch manufactured by

Thickness of plate cut ½ in Length of cut 20 ft Tip No. 1

Pressure at which gas was delivered to torchline:

Oxygen 25 lb per sq. in. Acetylene 3.02 lb per sq. in.

Time	Poten	Pres. at Reg. Oxygen	Pres. at Reg. Acety	Flow meter differentials Oxygen	Flow meter differentials Acety	Time	Poten	Pres at Reg Oxygen	Pres at Reg Acety.	Flow Meter differentials Oxygen	Flow Meter differentials Acety.
1	80	25.00	3.00	55.4	32.2						
3		25.05	2.90	52.3	32.3						
		Flame readjusted									
5		24.95	3.05	53.2	26.6						
7		25.00	2.98	51.4	28.4						
		Flame readjusted									
		Flame readjusted									
9		25.00	3.20	55.0	19.2						
11		25.00	3.00	52.0	14.0						
13	80	25.00	3.00	52.1	17.4						
Mean		25.00	3.02	53.1	24.3						

FIG. 29.—*Log sheet No. 6, gage board—cutting*

Date 5-27-20
Computed by T.W.G.
Checked by E.R.B.

Sheet No. 7

Computations
Test No. 6-a

		Oxygen	Acetylene
Mean Line Pressure.	Lb. per Sq. In.	25.00	3.02
Line Drop.	Lb. per Sq. In.	.20	.04
Mean Torch Pressure.	Lb. per Sq. In.	24.70	2.98
Mean Atmosphere.	Lb. per Sq. In.	14.56	
Value of Flash Back Head.	Lb. per Sq. In.	.00	.10
Absolute Flow Meter Pressure.	Lb. per Sq. In.	39.56	17.68
Mean Potentiometer Reading.	Millivolts	.80	.82
Mean Temperature Factor.		.999	1.003
Weight of Gas Used.	Pounds	1.095	.179
Density of Gas.	Lb. per Cu. Ft.	.0890	.0743
Volume of Gas by Balance.	Cubic Feet	12.30	2.41
Mean Rate of Flow by Flow Meter.	Cu. Ft. per Hr.	52.5	10.2
Duration of Test .232 Hours			
Volume of Gas by Flow Meter.	Cubic Feet	12.18	2.37
Loss of Tank Pressure.	Lb. per Sq. In.	15	2
Cubic Feet per Pound of Tank Pressure.		—	—
Volume of Gas by Test Gage.	Cubic Feet	—	—
Volume Ratio by Balance.		—	
Volume Ratio by Flow Meter.		—	
Length of ~~Weld~~ or Cut in Feet.		20	
Linear Feet per Hour ~~Welded~~ or Cut.		86	
Mean Tank Pressure.	Lb. per Sq. In.	261	

FIG. 30.—*Log sheet No. 7, computations—cutting*

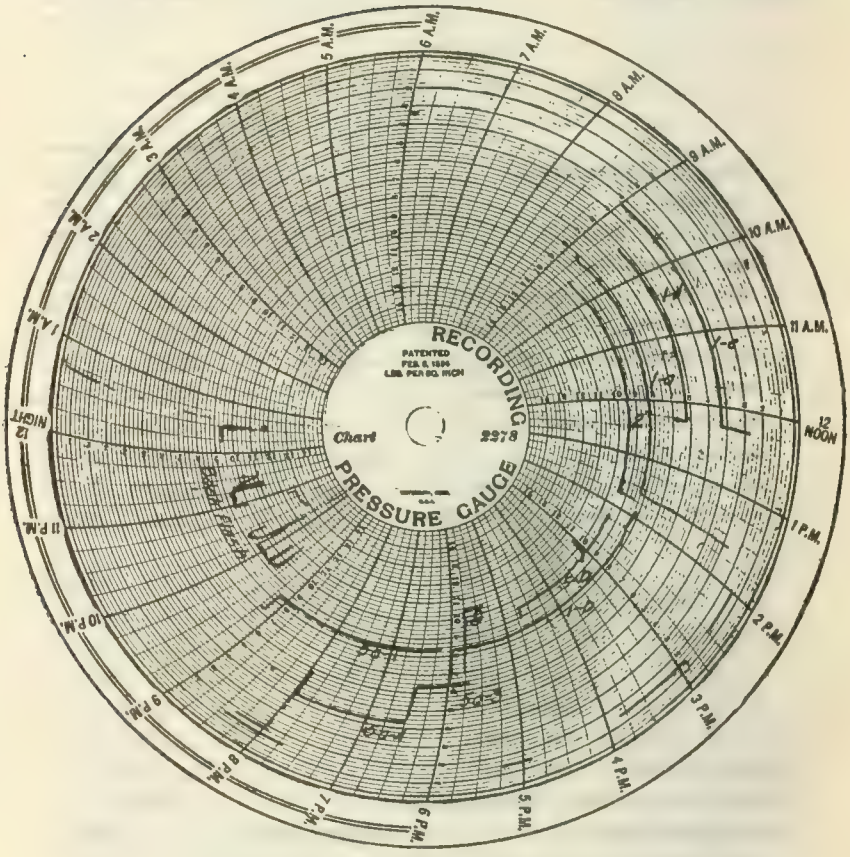


FIG. 31.—*Pressure gage record showing closeness of pressure control*

For certain studies on the probable economies of operation of the various blowpipes it was necessary to assume certain fixed values as units of compensation and cost. While this procedure may be open to certain criticisms due to the fact that the assigned values are arbitrary and may not apply under conditions of operation ordinarily met with in commercial work, still it is only through such economy studies that certain features of blowpipe operation can be distinctly shown. For such studies the standard salary schedule of the United States Naval Gun Factory, Washington, D. C., for the summer of 1920 was selected as the basis of compensation for labor costs. On this assumption the labor costs for welding and cutting have been computed at 80 cents per man-hour. For specifying the value of the gas consumed during any particular operation the prices of $1.50 per hundred cubic feet for tanked oxygen and $2.75 per hundred cubic feet for tanked acetylene were assumed. The prices are those at which such gases were furnished to the Bureau and seem on the whole to represent general average costs for such commodities, exclusive of shipment charges. An unsuccessful attempt was made to secure what might be considered general average prices from several of the larger gas manufacturers. Before definitely settling on the above prices, cost data for several of the test series were computed for varying ratios of cost of oxygen to acetylene. These latter figures showed that any reasonable change in the basis of cost caused only a few unimportant changes in the relative rating of the blowpipes.

In the computations of cost of operation only gas and labor costs are included. No additions have been made for maintenance of equipment, overhead charges, and secondary supplies. It is felt that this procedure furnishes a satisfactory basis of analysis, as external conditions and management will largely control the value of secondary costs and vitiate any assumptions made concerning the value of such items.

Certain of the plotted data mentioned above contain curves showing the relative velocity of exit of gases from the blowpipe tips. In order to plot these curves for the cutting blowpipes it was assumed that the volume of oxygen consumed by the preheating flames was equal to the measured volume of acetylene consumed, and that the volume of oxygen available for the oxygen cutting jet was therefore the total measured volume of

oxygen consumed, minus an amount equal to the measured volume of acetylene consumed. For the welding blowpipes the velocity of exit was based upon total volume of both gases consumed.

2. THE CUTTING BLOWPIPE

The summaries of the data obtained from cutting tests made with the various blowpipes are shown in Figs. 32 to 41 and Table 3. The curves of Figs. 32 to 35 are plotted to show, for the thickness of metal used for the tests, the relations existing between the volumes of oxygen and acetylene consumed per hour, the linear feet of metal cut per hour, and the velocity of exit of the oxygen cutting jet. A study of these curves shows that the rate of cutting apparently increases with the volume of oxygen consumed. In these curves the blowpipes are arranged in the order of the oxygen consumption.

For determining the relative economy of operation the graphs of Figs. 36 to 39 have been plotted to show the cost per linear foot of cut for each blowpipe. In these curves the blowpipes have been arranged in the order of the cost of operation.

The study of these data shows some rather interesting conditions relative to the art of metal cutting by the oxyacetylene process:

1. The very great irregularity in the shape of the various graphs indicates quite clearly that if the individual blowpipes have been designed according to a theory there does not exist any generally accepted theory. It is interesting to note in this connection that in cutting identical material under standardized conditions the various blowpipes exhibit, as indicated by the data for cutting ½ inch metal (Fig. 32), variations in speed of cutting from 33 to 109 feet per hour, or 330 per cent; variations in volume of oxygen consumed from 30 to 130 cubic feet per hour, or 430 per cent; variations in velocity of exit of cutting jet oxygen from 530 to 1975 feet per second, or 390 per cent; and variations in volume of acetylene consumed from 7 to 21 cubic feet per hour, or 300 per cent.

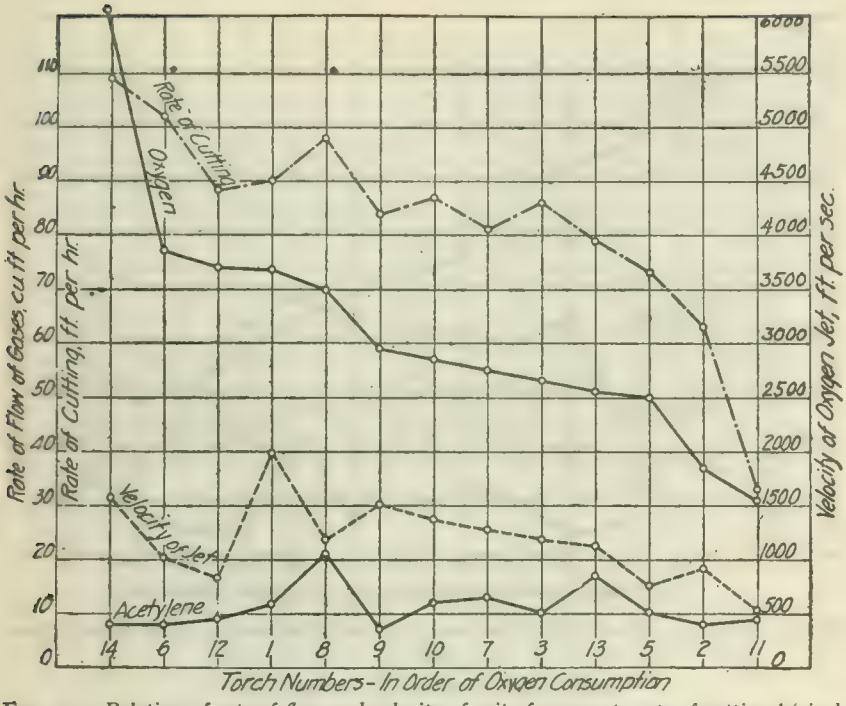

Rate of Flow of Gases, cu ft per hr.
Rate of Cutting, ft. per hr.
Velocity of Oxygen Jet, ft. per sec.
Rate of Cutting
Oxygen
Velocity of Jet
Acetylene
Torch Numbers - In Order of Oxygen Consumption
14 6 12 1 8 9 10 7 3 13 5 2 11 0

FIG. 32.—*Relation of rate of flow and velocity of exit of oxygen to rate of cutting ½-inch metal*

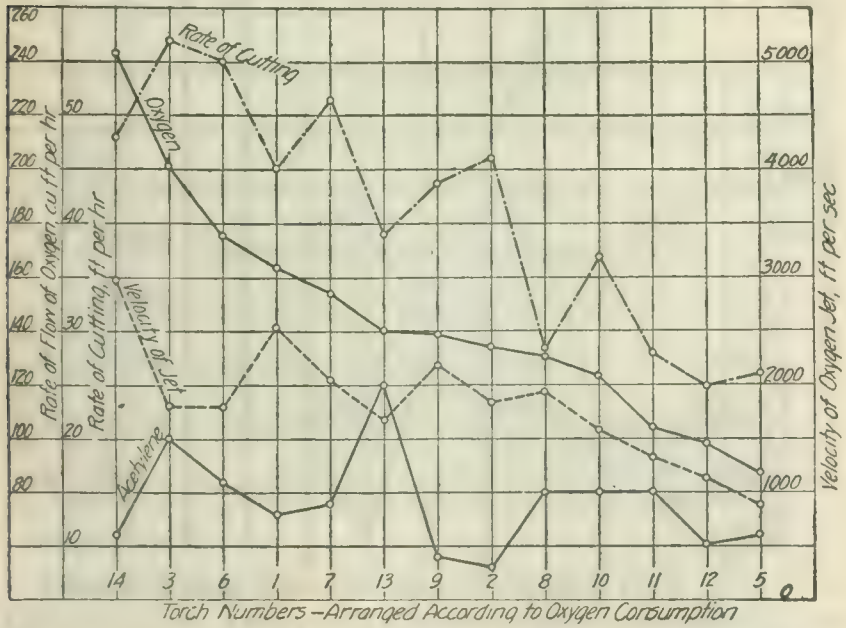

Rate of Flow of Oxygen, cu ft per hr
Rate of Cutting, ft per hr
Velocity of Oxygen Jet, ft per sec
Rate of Cutting
Oxygen
Velocity of Jet
Acetylene
Torch Numbers – Arranged According to Oxygen Consumption
14 3 6 1 7 13 9 2 8 10 11 12 5 0

FIG. 33.—*Relation of rate of flow and velocity of exit of oxygen to rate of cutting 2-inch metal*

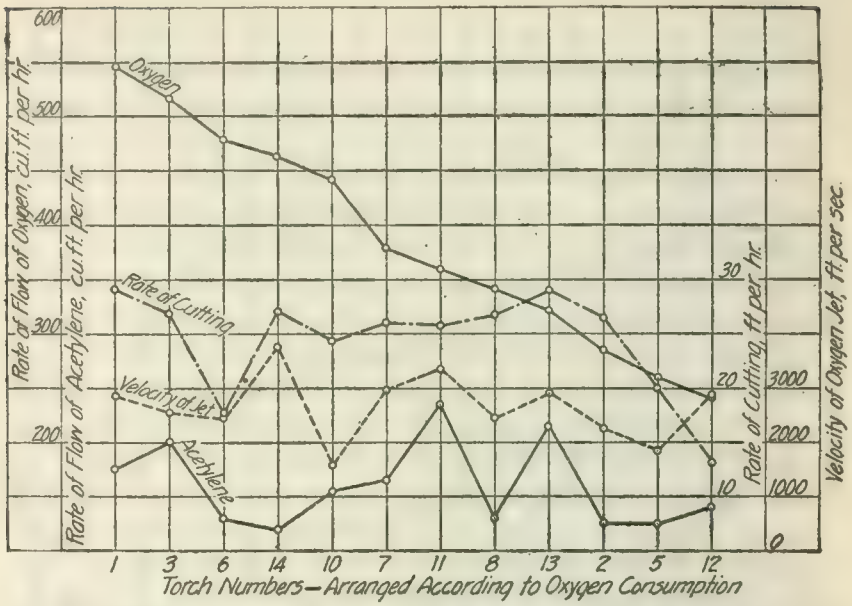


FIG. 34.—*Relation of rate of flow and velocity of exit of oxygen to rate of cutting 6-inch metal*

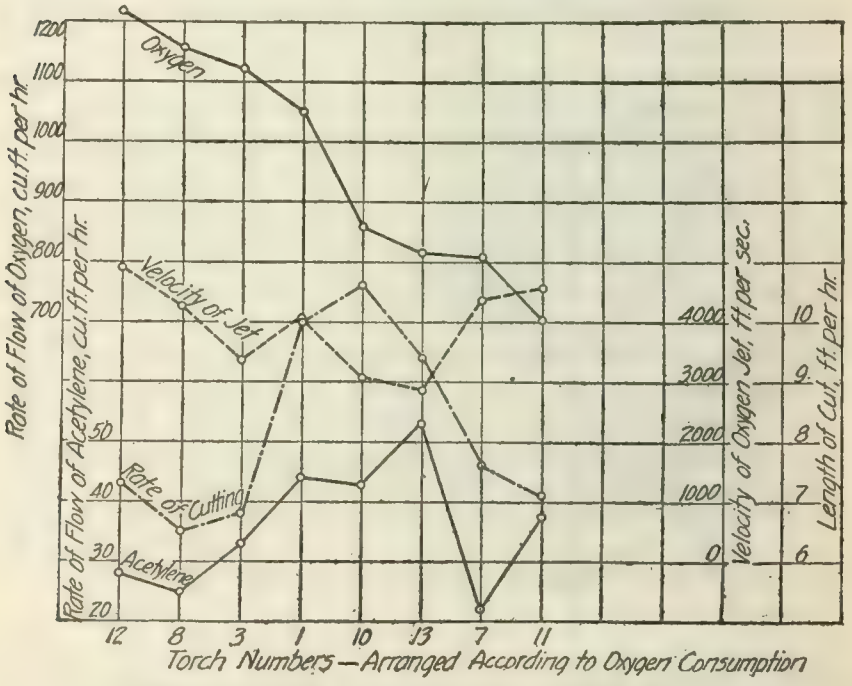


FIG. 35. *Relation of rate of flow and velocity of exit of oxygen to rate of cutting* **10-inch** *meta l(see footnote d, Table 3)*

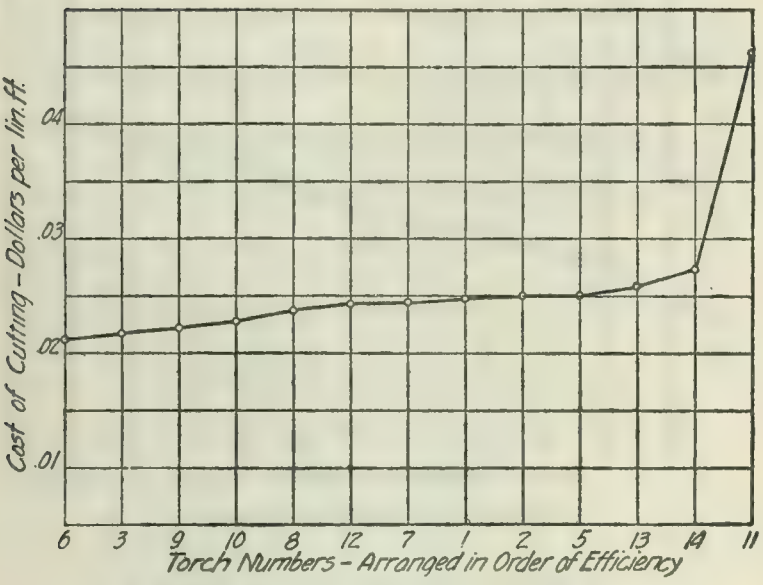


FIG. 36.—*Cost of cutting ½-inch metal*

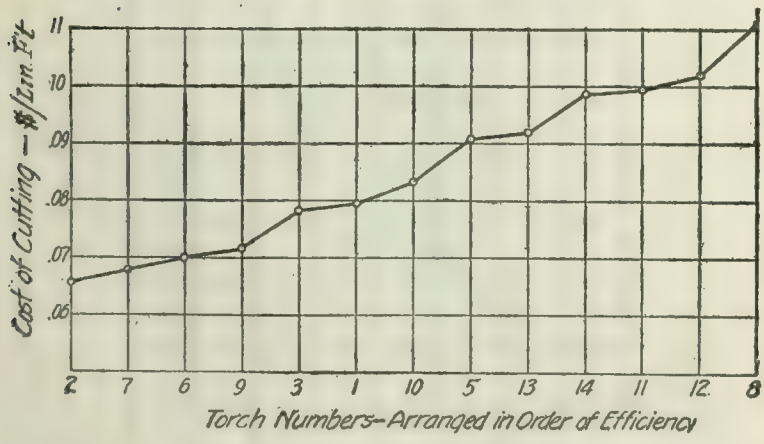


FIG. 37.—*Cost of cutting 2-inch metal*

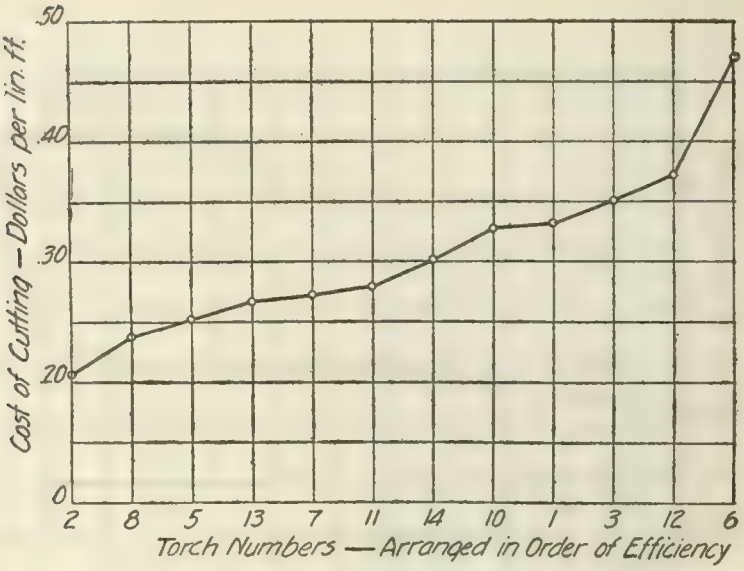


FIG. 38.—*Cost of cutting 6-inch metal*

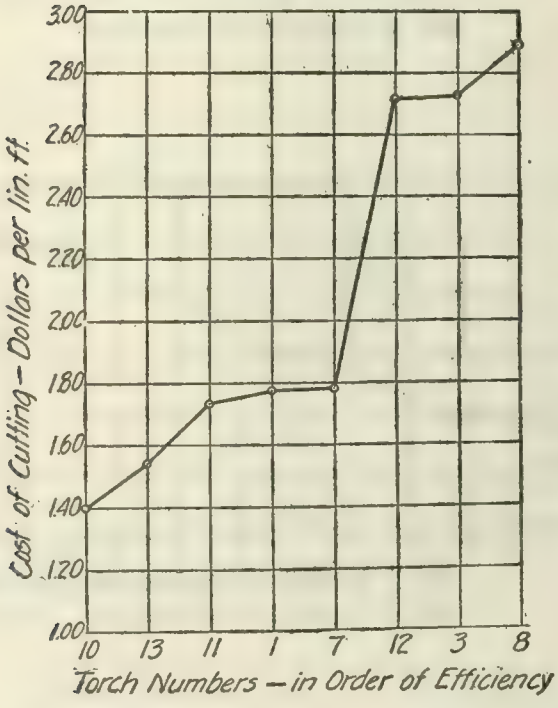


FIG. 39.—*Cost of cutting 10-inch metal (see footnote d, Table 3)*

TABLE 3.—General Summary of Cutting-Torch Data

Test 6—¼-inch material

Torch No.	Tip No.	Pressure delivered at torch handle		Length cut per hour	Volume consumption		Velocity of oxygen jet	Length cut per cubic foot of oxygen	Cost per linear foot cut
		Oxygen	Acetylene		Oxygen	Acetylene			
		Lbs./in.²	Lbs./in.²	Feet	Ft.³/hr.	Ft.³/hr.	Ft./sec.	Feet	Dollars
[a](1)	[a](2)	[a](3)	[a](4)	[a](5)	[a](6)	[a](7)	[a](8)	[a](9)	[a](10)
1	2	19.32	3.95	90	74	12	1975	1.45	0.025
2	1	14.79	2.98	63	37	8	922	2.17	.025
3	1	24.70	2.98	86	53	10	1184	2.00	.022
4[b]									
5	2	9.53	4.97	73	50	10	752	1.82	.025
6	2	19.21	2.98	102	77	8	1010	1.48	.021
7	1	29.69	4.94	81	55	13	1272	1.93	.024
8	1	24.52	5.90	98	70	20	1180	2.00	.024
9	3	39.74	4.98	84	59	7	1502	1.61	.022
10	1	24.68	6.97	87	57	12	1364	1.93	.023
11	1	5.74	1.94	33	31	9	530	1.50	.046
12	2	11.04	1.99	[illegible]	74	9	834	1.35	.024
13	1	24.82	.88	79	51	17	1139	2.32	.026
14	2	28.34	1.72	109	130	8	1564	.89	.027

Test 7—2-inch material

1	2	56.05	3.96	45	163	13	2523	0.300	0.079
2	2	42.70	2.97	46	134	8	1842	.366	.066
3	3	37.10	4.86	57	210	20	1814	.300	.078
4[b]									
5	3	23.08	4.95	26	87	11	863	.342	.091
6	[illegible]	48.09	3.89	55	175	16	1805	.346	.070
7	2	53.70	4.93	51	154	14	2050	.364	.068
8	2	44.83	3.91	28	130	15	1934	.243	.111
9	[illegible]	58.97	5.99	44	139	[illegible]	2188	.338	.072
10	2	33.84	4.92	37	123	15	1580	.342	.083
11	[illegible]	28.95	2.90	28	104	15	1302	.314	.099
12	[illegible]	21.91	2.97	25	98	10	1129	.284	.102
13	2	44.26	.72	39	140	25	1683	.339	.092
14	3	45.91	1.95	48	243	11	2979	.207	.099

Test 8—6-inch material

1	4	74.70	4.02	29	546	25	2865	0.056	0.332
2	[illegible]	72.60	4.91	27	285	15	2260	.100	.206
3	[illegible]	75.83	5.74	27	517	30	2532	.055	.350
4[b]									
5	4	[illegible]	6.96	20	256	15	1824	.083	.253
6	5	82.20	3.91	18	479	16	2455	.039	.478
7	3	90.60	9.88	26	378	23	2970	.073	.273
8	3	69.99	3.90	27	342	16	2466	.083	.238
9[c]									
10	[illegible]	72.38	3.84	24	441	21	1589	.057	.327
11	3	103.20	6.83	26	360	36	3365	.080	.280
12	4	67.39	3.63	13	241	18	2861	.058	.372
13	3	79.30	.25	29	388	41	2904	.084	.267
14	4	70.10	3.42	27	463	14	3761	.060	.301

[a] Numbers in parentheses are column numbers.
[b] Not submitted to cutting tests.
[c] Torch failed to cut in this test.

TABLE 3—Continued

Test 9—10-inch material

Torch No.	Tip No.	Pressure delivered at torch handle		Length cut per hour	Volume consumption		Velocity of oxygen jet	Length cut per cubic foot of oxygen	Cost per linear foot cut
		Oxygen	Acetylene		Oxygen	Acetylene			
		Lbs./in.²	Lbs./in.²	Feet	Ft.³/hr.	Ft.³/hr.	Ft./sec.	Feet	Dollars
[a] (1)	[a] (2)	[a] (3)	[a] (4)	[a] (5)	[a] (6)	[a] (7)	[a] (8)	[a] (9)	[a] (10)
1	5	129.80	5.73	10.0	1050	44	4012	0.0100	1.776
2 [b]									
3	5	123.90	7.68	6.8	1122	33	3385	.0064	2.725
4 [c]									
5 [b]									
6 [b]									
7	4	153.00	11.88	7.6	809	22	4350	.0102	1.780
8	4	165.70	3.79	6.5	1157	25	4281	.0053	2.898
9 [b]									
10	4	152.90	14.65	10.6	858	43	3085	.0135	1.400
11	4	164.17	4.63	7.1	701	37	4565	.0106	1.737
12	4	160.24	5.73	7.3	1216	28	4912	.0058	2.716
13	4	117.32	—.69	9.4	816	53	2889	.0117	1.542
14 [b] [d]									

[a] Number in parentheses are column numbers.
[b] Torch failed to cut in this test.
[c] Not submitted to cutting tests.
[d] By inadvertent error a No. 4 tip was used instead of a No. 8*a* in making the 10-inch cut reported on above. On a previous test with an 8*a* tip torch No. 14 made a cut through 10-inch material using an oxygen velocity of 7310 feet per second; consuming 1340 cubic feet of oxygen and 18.55 cubic feet of acetylene per hour; cutting at the rate of 9.23 feet per hour at a cost of $2.32 per linear foot of cut, which is at the rate of 0.0067 foot of cut per cubic foot of oxygen. The cut was of fair quality.

2. In general the speed of cutting increases with the volume of oxygen consumed, provided the velocity of exit of the cutting jet and the amount of acetylene consumed in the preheating flame is kept constant.

3. With the same velocity of exit of cutting jet and equal oxygen consumption the speed of cutting is reduced and therefore the cost of operation increased by an increase in the amount of acetylene consumed in the preheating flames, the increased amount of acetylene requiring a larger amount of oxygen for its combustion and thus decreasing that available as an oxidizing agent in the cutting jet.

4. With equal acetylene consumption and equal oxygen consumption the rate of cutting and therefore the decrease in cost of operation increases with the velocity of exit of the oxygen-cutting jet, up to a certain limit. Beyond this limit a further increase does not further decrease the cost of operation.

5. With equal acetylene consumption, increasing the velocity of exit of the oxygen jet and at the same time the oxygen consumption does not necessarily increase the speed of cutting in commensurate ratio and may even tend toward increased cost of operation. It seems apparent that there is an economical limit to such increases, and that with too high a velocity an

increased oxygen volume may be wasted. This indicates that for any thickness of metal there is a limiting velocity at which complete utilization of the oxygen takes place and a limiting value to the amount of oxygen required to produce the cut.

As indicated above there is a great deal of difference between the characteristics of blowpipes of different designs. One of the most interesting and noticeable differences is the fact that no make is equally proficient and economical for all thicknesses of metal. Of the six most economical cutting blowpipes on ½-inch material, two failed to cut 10-inch material, and the remainder stand almost in the inverse order of rating. Further, two of the most uneconomical blowpipes on ½-inch material are two of the most economical on the heavy or 10-inch cutting. It is interesting to note that this reversal of performance in general seems to be progressive with the variation in thickness of metal cut. (See Table 3 and Figs. 36 to 39.)

The reason for reversal of performance, upon examination of the data available, seems to be quite clearly indicated. The average design of blowpipe is operated with the oxygen jet issuing from the tip at too great a velocity. It will be noted that the most efficient blowpipes for the metal of various thicknesses are those operating at the lower velocities. Where there appears to be a divergence from this fact it will be further noted that those blowpipes seeming in performance to be inconsistent with the above statement are those using an excessive amount of acetylene.

A portion of the data of Table 3 replotted to show the relation between area of metal cut per cubic foot of oxygen consumed for the various thicknesses (See Fig. 40) is of further interest, as it suggests that none of the blowpipes submitted to test are as economical in cutting 2-inch and possibly 10-inch metal as might be expected. The data of Fig. 40 show distinctly that the amount of metal cut per cubic foot of oxygen was greatest for the ½-inch material. Further, if the oxygen in all cases could be used with the same economy, and the same width of kerf as obtained with the ½-inch material secured for metal of various thicknesses, the relation between square inches of metal cut per cubic foot of oxygen used and the various thicknesses would lie on a horizontal straight line (*a–a* Fig. 40) through the point of maximum efficiency for the ½-inch material. The replotted data, however, lie below this straight line.

It should be expected that in reality the width of kerf would increase with the thickness being cut, and on this basis it is evident

that a curve representing the relation between the area of metal cut per unit volume of oxygen consumed and the thickness would be a smooth regular curve falling constantly further below the line *a–a* with the increased thickness of metal cut. Further, such a curve would either approach parallelism with the horizontal axis or actually cut it at some definite point.

The law governing this relation has not been determined during this investigation. It is possible that there is a certain minimum kerf required for cutting thin metal, and accepting this with the further assumption that beyond this minimum kerf the width of kerf increases linearly with the thickness cut, the law of variation

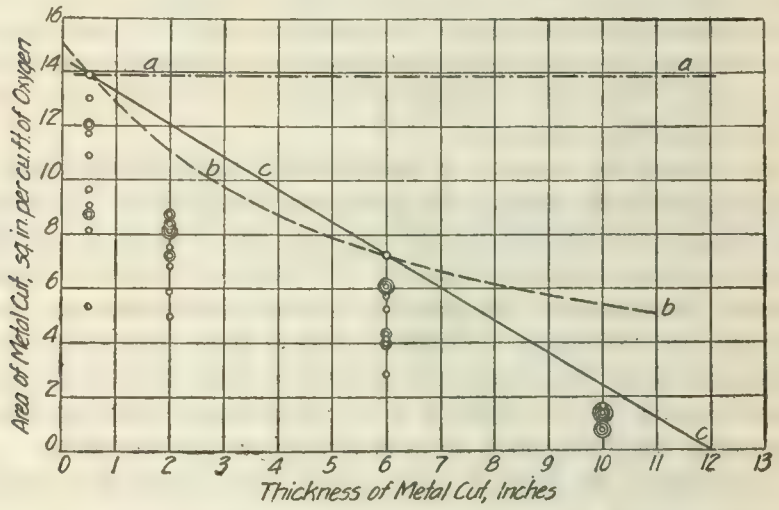


FIG. 40.—*Variations in efficiency of blowpipes for cutting metal of different thicknesses*

between area of metal cut for unit volume oxygen consumption and the thickness of metal cut would be expressed by a curve of the form indicated by the line *b–b*. If further it is assumed that the cuts for ½ and 6 inch metal were probably as nearly as economical as it is possible to secure them, this curve would actually occupy the position at which it is shown on the sheet.

A more simple assumption can, however, be made—that is, that the above relation is expressed by a sloping straight line, and it is of interest to note that such a line *c–c* can actually be passed nearly through all the points of maximum economy plotted on the sheet.

On either of these assumptions it is very evident that none of the blowpipes tested are cutting 2-inch material economically. It would seem that the reason for this noneconomy of operation

is probably due to two things. First, the use of an oxygen cutting jet issuing at too great a velocity. If a velocity of 2200 feet per second is sufficient for cutting 6-inch material, it would appear that a velocity of 1800 feet per second is excessive for 2-inch material. Secondly, in the blowpipes tested the preheating flames in most cases were too large for the work required. In fact, in a number of instances there was practically very little difference between the size of the preheating flame used for the 2-inch plate and that used for the 6-inch material. For the thinner material the large-sized preheating flame would tend toward noneconomy in operation inasmuch as it would preheat a relatively large surface, with the resulting tendency for the oxygen to oxidize the entire body of metal thus preheated, giving probably a wider kerf than necessary and thus using the oxygen supply nonadvantageously. This conclusion is further substantiated by the fact that in cutting 2-inch material practically none of the slags showed metallic characteristics but were on the other hand fully oxidized metal.

A further study of the data presented in Fig. 40 indicates also that possibly the maximum economy has not been fully secured for the 10-inch material. It is certain, though, in this connection that the relation between metal cut per unit volume of oxygen consumed and metal of different thicknesses can not follow the law suggested and represented by the curve *b–b*, for this curve implies that, in addition to a possible increase in economy over that actually obtained, far greater thicknesses could be cut with relatively slight changes in economy. This is contrary to known facts and is rather forcibly contradicted by the appearance and character of the cuts made on 10-inch material and shown in Figs. 44 to 46, inclusive. It seems probable, then, that the true economy relation is more nearly represented by the straight line *c–c*. If this is true, then a slight improvement might be expected in cutting 10-inch material economically. On the other hand, such a conclusion leads directly to the fact that the maximum available cut of oxyacetylene blowpipes of present design lies in the vicinity of 12 inches. As an interesting side light on this, Fig. 47 shows an attempted cut for maximum thickness made with the blowpipe that exhibited maximum efficiency in heavy cutting. The cut penetrated about 12 inches into a 16-inch block, the bottom of the cut being practically at right angles to the direction of cut.

This immediately brings up the question as to why the limit of depth of cut for an oxyacetylene blowpipe should apparently be about 12 inches. It has been known that in heavy cutting the cleaner cuts and greater depths were secured with the oxyhydro-

gen blowpipe. (See Fig. 46*B*.) The reason generally given for this difference is that the oxyhydrogen preheating flame is longer and therefore secures better preheat penetration within the cut. It would seem, however, that a possibly more probable explanation of the difference lies in the resulting products of combustion and that this is also a possible explanation of the limitation of the oxyacetylene blowpipe in heavy cutting. Besides the iron oxide, water vapor and carbon dioxide are formed during the cutting process. With the use of acetylene instead of hydrogen a greater proportion of carbon dioxide is formed. This gas sinking to and being partially trapped in the lower portion of a cut and being a nonsupporter of combustion causes a slackening of the oxidation process and in deep cuts it may be present in sufficient volume to actually arrest the oxidation process. This suggestion would seem to be verified by the experience that in deep cutting a large increase in the kinetic energy of the issuing cutting jet produced by increasing the pressure at which the gas is used will often secure a cut where the blowpipe would otherwise not have been enabled to do so. The increased pressure and resulting energy insures that the products of combustion are blown out of the way. Under such conditions increased depths of cuts may be secured, but not necessarily economically.

Unfortunately these questions have not been definitely settled by these tests and the above must therefore be looked upon simply as indications and suggestions for further work toward securing increased economy in cutting.

In questions of economy of operation the volume of gas used plays an important part. Where an excessive amount of acetylene is used, acetylene being the more expensive gas, the economy of operation is affected adversely. This question of acetylene consumption is one that would seem to deserve attention. The size of the preheating flames is a direct index of the acetylene consumption. As indicated above in a great many of the blowpipes tested, it was found necessary to reduce the acetylene pressure under that specified by the manufacturer (thereby reducing the size of the flame or acetylene consumption) in order to secure the flame size that did not melt too much of the top edge of the cut and reunite the pieces cut apart by the flow of this molten metal back into the kerf. In some instances with the specified preheating flame the pieces were practically welded together again along the top edge, and in others the top edge was badly beveled.

Neither of the above conditions, developed by excessive sized preheating flames, is desirable. Even though not welded together a badly beaded top edge (Fig. 43*H*) or a badly beveled top edge would in most cases increase the cost of following manufacturing operations. The feeling of the blowpipe manufacturers on this matter seems to be that they must have excessive preheating flames to take care of heavy scale, slag, etc., and to facilitate rapid cutting, especially at the start of a cutting operation. The experience of this investigation as developed on cutting does not indicate that a cut can be started any quicker with an excessive flame than with one of proper size. In cases where slag or heavy scale are met with it is doubted whether burning slag and scale

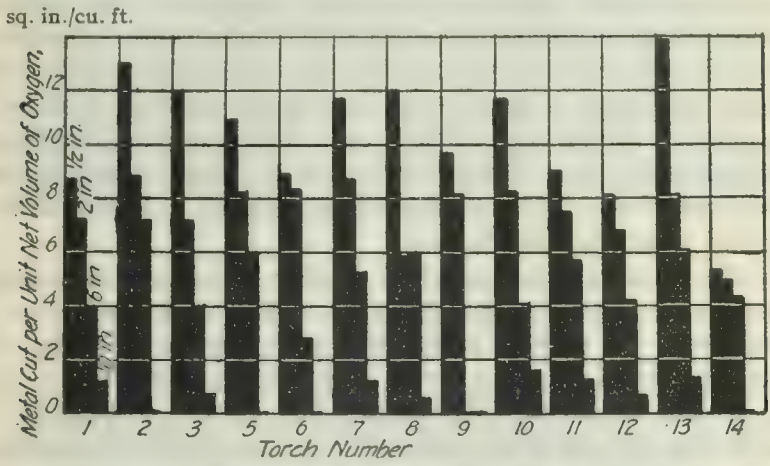


FIG. 41.—*Relation of volume of oxygen used to area of metal cut for metals of varying thicknesses*

off at an oxygen cost of $1.50 to $10 an hour (according to thickness of material cut) can economically compete with unskilled hand labor (chisel and hammer) at 75 cents per hour.

The quality of cut secured is quite important. Figs. 42 and 43 are included to show what may be expected from a properly operating cutting blowpipe. Figs. 44 to 46*A* attempted cuts on 10-inch material, in this case armor plate, speak for themselves, and they may well be taken to indicate that none of the blowpipes tested are suitable for working to close limits on cuts of such depths. For comparison an oxyhydrogen cut on the same block is shown in Fig. 46*B*.

On the other hand, there is splendid evidence that some blowpipes can do especially nice work when properly designed for the thickness being cut, in this case 6 inches. (Fig. 43*F*). A sec-

tional view through this cut is shown on the right-hand side of Fig. 48, *c*. Fig. 42 *B* to *E* inclusive shows typical cuts of 2-inch and ½-inch metal, and Fig. 42*A* is further evidence that there is such a thing as using too much oxygen in executing a cut.

Besides the top edge and face of a cut, the bottom edge is of interest, as its appearance is quite closely connected with the type of slag formed. A completely oxidized slag has a greater tendency to cling to the underside and if excess oxygen is present there is considerable undercutting which tends to increase the amount of slag and to make it adhere more firmly, with the result that a rough bottom edge is produced. By the proper proportioning of the oxygen volume it is apparent that a slightly metallic slag can be secured, the metallic element being due to the fact that in the presence of an insufficient oxygen volume some of the metal is blown out without being oxidized. A slightly metallic slag clings quite firmly to the bottom edge of the cut, but apparently possesses an advantage in that it may be removed in long pieces as shown in Fig. 43*G*, and leaves behind it, as seen, a very smooth sharp under edge. The section of the 6-inch cut shown in Fig. 43*F* had a slightly metallic slag.

In the use of the oxyacetylene cutting blowpipe with especial reference to manufacturing operations the question arises as to what effect the cutting action has upon the metal body. To determine this a section through oxyacetylene cuts of ½, 2, and 6 inch plate which will be designated hereafter as blocks A, B, and C, respectively, were examined microscopically to determine the nature and extent of the surface changes induced in the metal by the oxyacetylene flame used for cutting the blocks. The examination included a study of the macroscopic appearance of a cross section after suitable etching, as well as of the comparative microstructure of the central or unchanged portion of the blocks and of the surface metal. In the following macrographs (Fig. 48) the specimens are shown natural size, the surface having been etched with an aqueous solution of ammonium persulphate. The depth to which the material has been affected by the flame is readily detected.

All of the micrographs of the material were taken at a magnification of 100 diameters, after etching with 2 per cent alcoholic nitric acid. These have been designated as A, B, and C to correspond to the macrograph of the same material. The micrographs in Fig. 49, *A*, *B*, and *C* show the structural condition at the extreme edge of the section, that is, where the flame affected

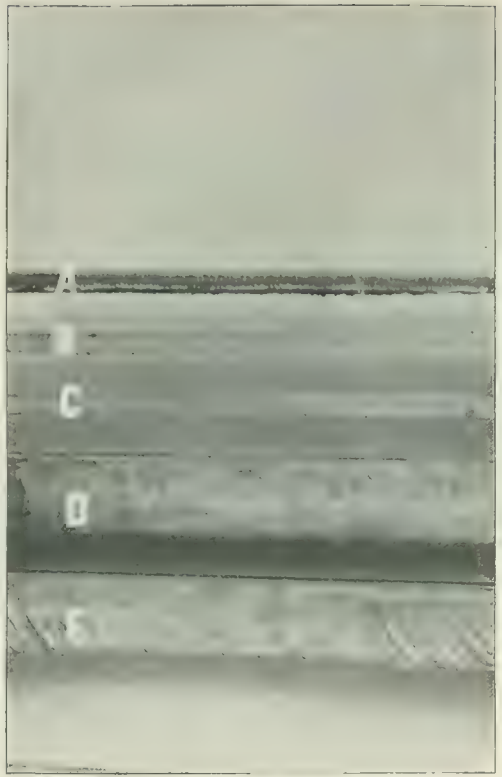


FIG. 42.—Oxyacetylene cuts on 1-inch and 2-inch steel

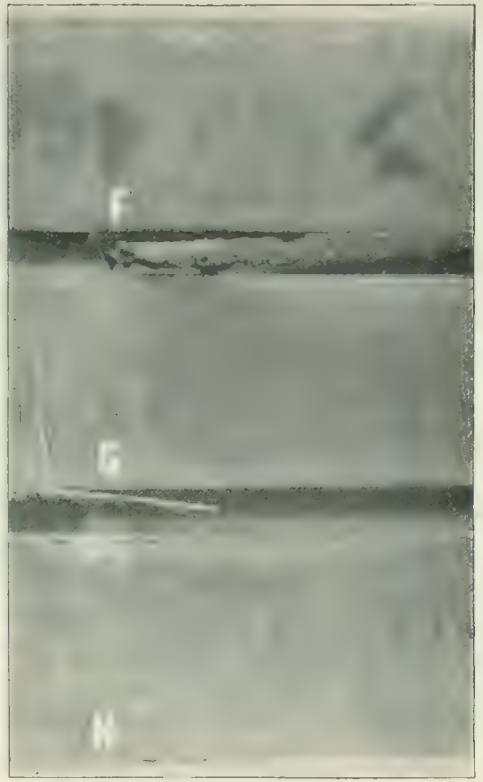


FIG. 43.—Oxyacetylene cuts on 6-inch steel

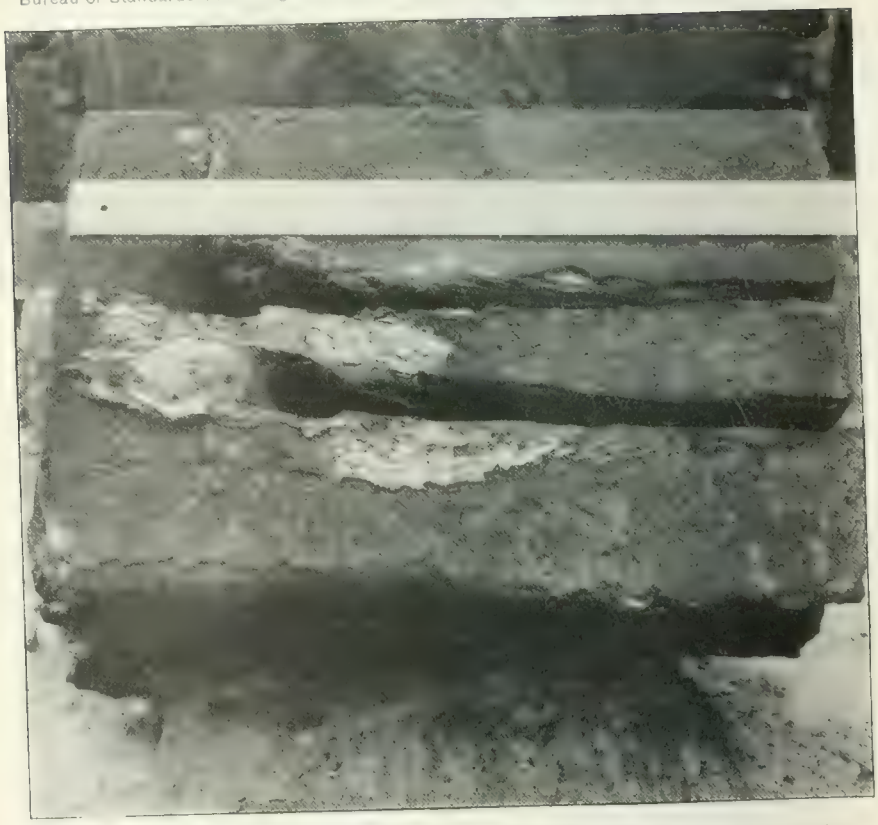


FIG. 47.—Oxyacetylene cut on 1-inch steel showing failure of cut at depth of 1 inches

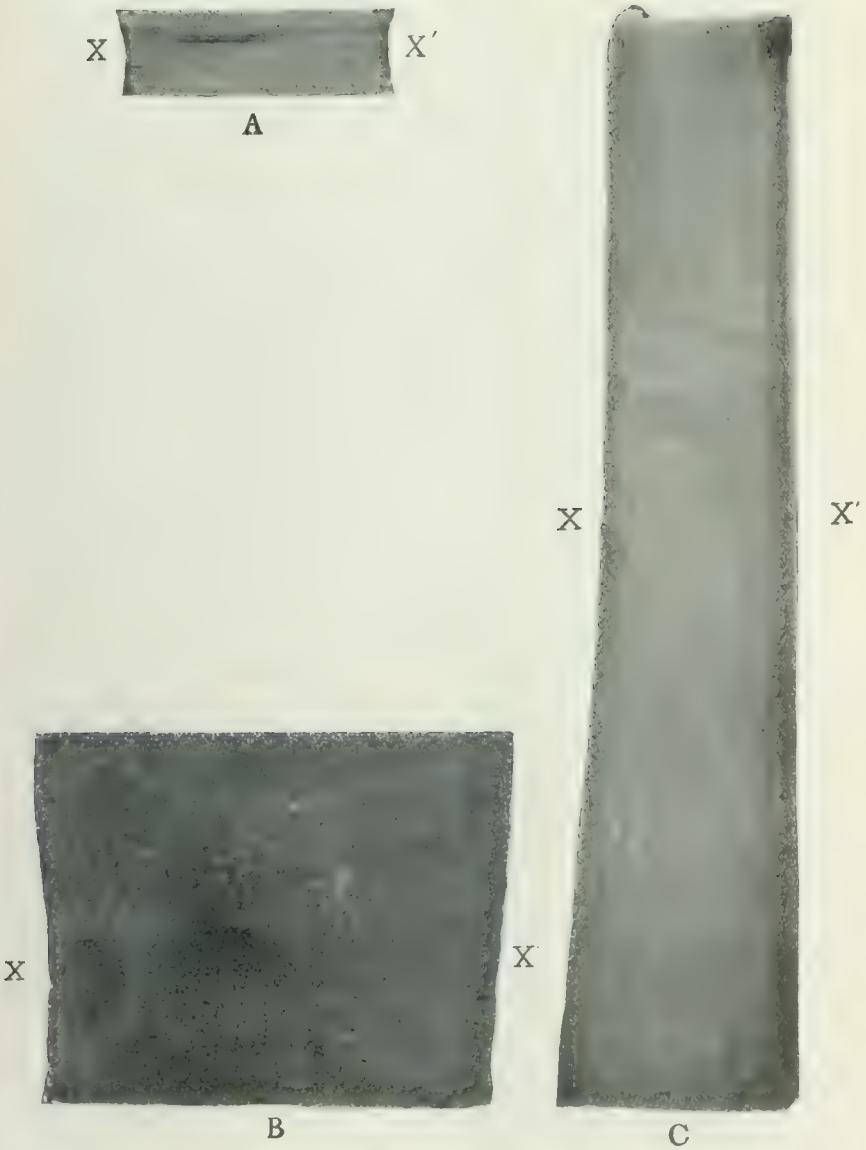


FIG. 48.—Macrographs of [illegible] inch metal (A), 2-inch metal (B), and 6-inch metal (C) cut during tests. X and X′ are the sides exposed to the flame.

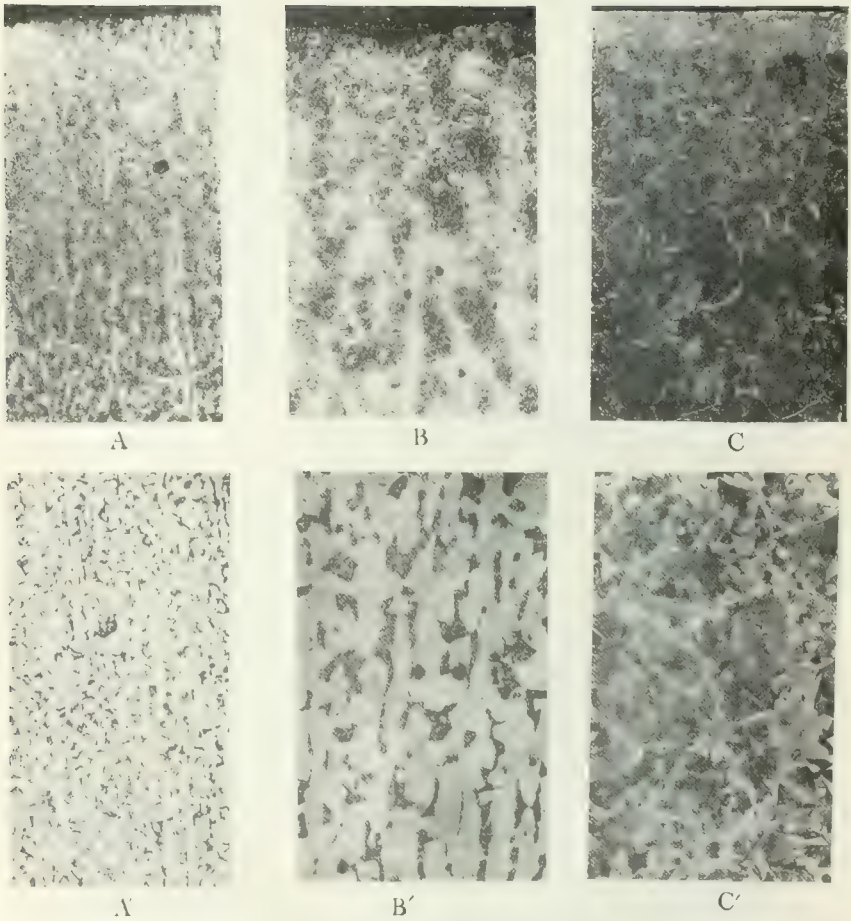
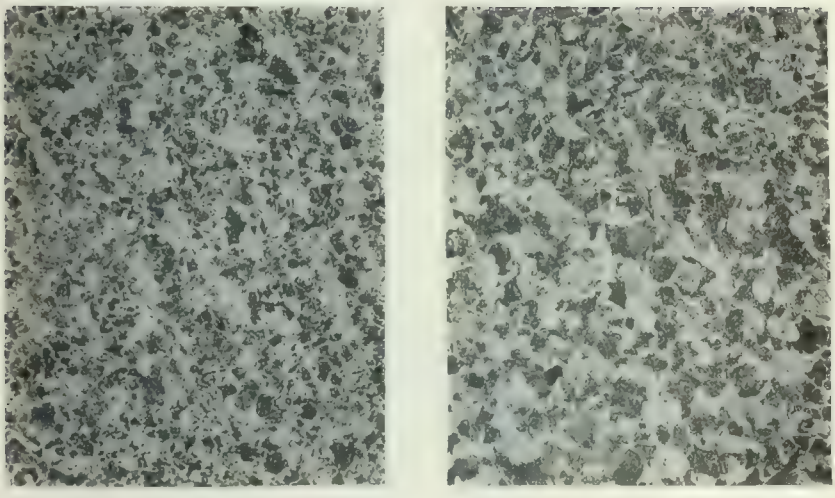


FIG. 49 A, B, C.—*Surface exposed to the oxyacetylene flame*

FIG. 49 A′, B′, C′. *Unchanged original structure of central portion of specimen*



C C′

FIG. 50.—*Micrographs showing the structure of the outer C and central C′ portions of the specimen which seemed to be most affected by the cutting process as shown in Fig. 49*

the metal, and *A'*, *B'*, and *C'* show the initial or unchanged metal of the central part of the block. The three specimens vary considerably in their carbon content, and the magnitude of the change in the surface metal of the specimen of highest carbon content (specimen C) has been much greater than in the others which were of lower carbon content.

In order to determine whether there had been appreciable carburization of the surface metal, such as the micrographs suggest, a sample of specimen C, which material was affected to the largest extent, was annealed at approximately 760° C, that is, above its critical temperature. Proper means were taken to avoid decarburization during the heating. The micrographs in Fig. 50 show the structure of the outer (affected) layer, *C*, and the central portion *C'*, at 100 diameters.

There appears to be no appreciable difference in the carbon content of the two parts of the annealed specimen. It may be concluded then that the appearance of the surface layer of the steel directly after cutting with the oxyacetylene flame is due to the chilling action of the large mass of unheated metal upon the hot surface after the flame has been removed. Such an effect would, of course, be more noticeable in a steel of higher carbon content than in one which is lower in this element. This of course accounts for the hard surface metal that is frequently encountered in working up metal cut with the oxyacetylene blowpipe.

A very noticeable feature in cutting blowpipe performance was the irregularity of action and the variation in length of the preheating flames and the frequency with which certain blowpipes developed flashbacks or went out during operations. In fact one particular and very well known blowpipe after six official test trials on ½-inch plate failed to make more than 43 feet of the required 50-foot test cut. As the causes back of these troubles are identical with the action of the welding blowpipes, discussion concerning them will be included in that on welding blowpipes.

Among the cutting tests there were included short lengths of cuts made on ½-inch plate with oxygen pressures set 50 and 25 per cent above the manufacturers' specified pressures and 25 and 50 per cent below them. As these tests, outside of a few inconsistencies which were attributed to the cutting operation being carried out on warm plate, furnish only check data on the conclusions noted above, they are presented for what they are worth without further discussion (Fig. 51).

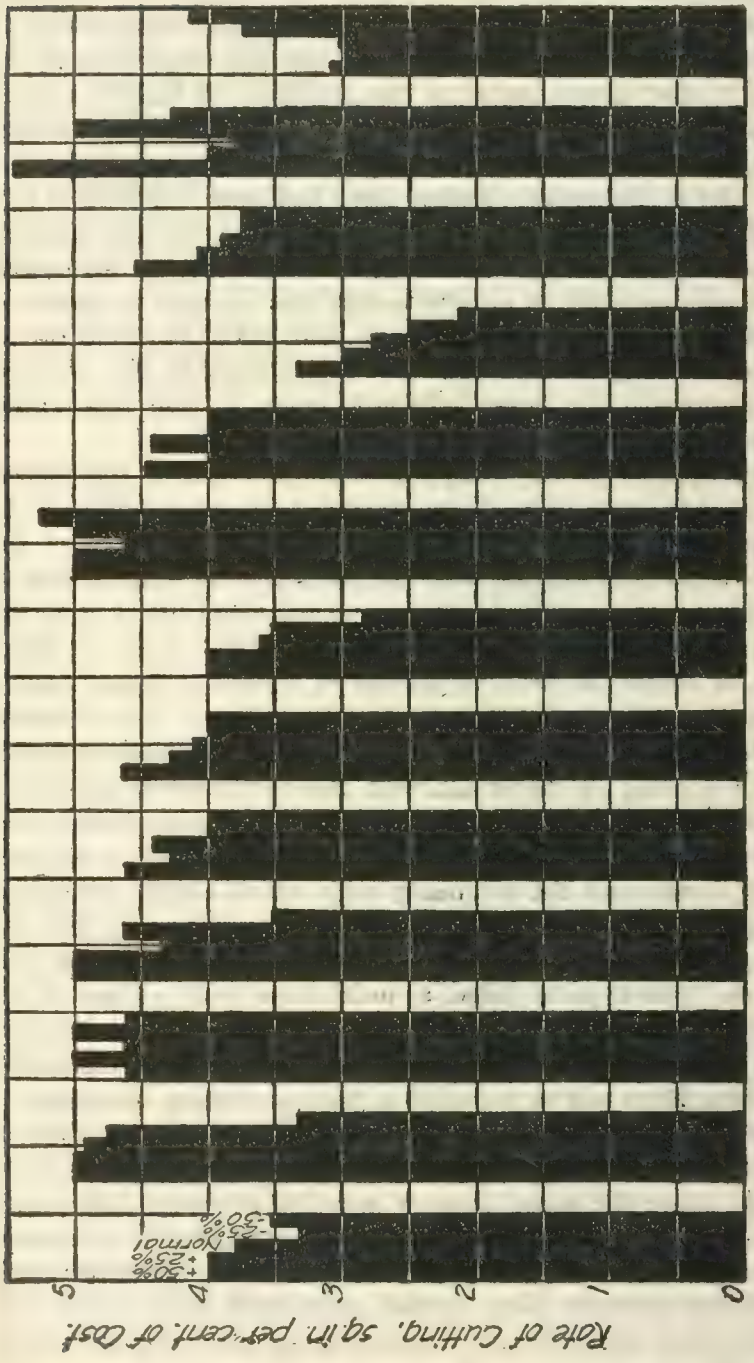


FIG. 51.—*Variations in cost of cutting 1/2-inch metal under varying pressures*

3. THE WELDING BLOWPIPE

It is universally accepted that outside of the mechanical features of design that affect weight, balance, and convenience of operation, the prime essentials of a strictly satisfactory piece of apparatus are:

1. Safety under all operating conditions.

2. Freedom from the so-called phenomenon of flash back or sustained back fire.

3. The quality of maintaining under all operating conditions a welding flame that is neither oxidizing nor carbonizing, one technically known as a "neutral flame," which in the process of combustion consumes, as nearly as possible, equal volumes of oxygen and acetylene; that is, maintains as nearly as possible the theoretical gas volume ratio of unity.

The tests of this investigation were decided upon with the idea of furnishing data that would enable blowpipe comparison relative to these essentials.

A study of the data obtained at the completion of the prescribed tests showed so many apparent inconsistencies that it was evident that there was a governing factor that was not understood, and that was, so far as test data were available, not in evidence. Irrespective of the fact that particular attention had been given to insure identical working conditions and gas pressure control, and that especial care was taken to secure exceedingly competent and unbiased operators, the results obtained from the welding tests seemed extremely unsatisfactory. Gas ratios obtained during actual welding operations were extremely high. Those obtained when the blowpipe was burning freely in air were also higher than was to be expected. In tests for flash backs there seemed to be a difference in the ease with which they could be developed in blowpipes of different manufacture, but there appeared to be no criterion that would enable one to say just why such phenomena could be caused more easily in some pieces of apparatus than in others, or why with some pieces of apparatus flash backs could be produced at times quite readily and at other times with difficulty. Finally, the general quality of the welds produced during test, although executed with the greatest care and, as shown by tensile tests, to be of a higher strength than is generally secured in most welding shops, was far from satisfactory.

It was evident that further information was essential if a satisfactory analysis of welding blowpipe performance was to be made. The greatest discrepancy from what was expected appeared to be in the high gas ratios obtained and the first attempt to answer the

problem was made by a study of gas-ratio phenomena. This seemed particularly desirable, as it has been very firmly held by almost all authorities that good welding can not be done with blowpipes having a high gas ratio.

An extensive series of supplementary gas-ratio tests was accordingly carried out. The results obtained accounted partly for the quality of the welding work secured. In some respects, though, they added increased confusion by lack of consistency.

In further study of the data secured by the prescribed tests, it was noticed that blowpipes that seemed especially susceptible to the phenomenon known as flash back were those in which the oxygen was delivered to the blowpipe at a pressure very much in excess of that at which the acetylene was delivered. It was further noticed that even among such blowpipes inconsistencies appeared. Critical examination of tip design in these later suggested a possible explanation. On this basis another series of supplementary tests was made and from the results of these data secured that clearly explain the cause of flash-back phenomena.

The logical continuance of the theory evolved for the cause of the flash-back phenomenon leads directly to the question of safety in operation and correct gas ratio and will explain, in large part, the reason for so large a proportion of oxyacetylene welds being of inferior grade.

The essential qualifications for a satisfactory welding blowpipe as enumerated above are therefore very intimately connected with the conditions governing the phenomenon of flash back. It seems desirable then to begin the discussion of the results obtained in the tests of the investigation by the critical analysis of the conditions conducive to the development of so-called flash backs in welding blowpipes.

Before taking up this discussion it will be of value to define the term "flash back" or, as it is sometimes called, "sustained back fire." It was very evident throughout the investigational tests that the representatives of the manufacturers were not in agreement as to what constitutes a flash back. To some the terms flash back and "back-firing" were synonymous, others distinguished between the blowpipe action representing these terms, and at least one other manufacturer attempted to distinguish blowpipe behavior by incorporating a term "preignition." There is therefore an apparent confusion of ideas.

For the purpose of this discussion the following definitions have been accepted as representative of blowpipe behavior under certain operating conditions.

1. Back Fire or Preignition.—Momentary retrogression of the blowpipe flame into the blowpipe tip which may relight immediately upon withdrawing the blowpipe away from the test piece or necessitate the reignition of the gases by means of a lighter. In this latter condition the flame will ignite properly without manipulation of the blowpipe handle valves.

2. Flash Back or Sustained Back Fire.—The retrogression of the burning flame back into the mixing chamber accompanied by the well-known marked hissing or squealing sound and the characteristic smoky sharp pointed flame of small volume necessitating the immediate cutting off of the gas supply by means of the blowpipe handle valves to prevent severe heating and possible destruction of the blowpipe head, and generally necessitating cooling the head before the blowpipe can be reignited.

The desirability of securing a nonflash back blowpipe has generally been considered to be due to its freedom from inconvenience and loss of time during welding operations and to its safety. Probably nothing is more detrimental to the poise of an operator and his desire to perform excellent work than an instrument that is constantly giving trouble. But the principal detriment in a blowpipe that flashes back, other than that of loss of time during welding operations, is in the possibility that a flash back may propagate itself back into the gas lines and in so doing possibly become responsible for a violent explosion with its accompanying dangers.

Blowpipe designers have devoted considerable time and money to the problem of minimizing or possibly obviating this latter danger and have devised a number of blowpipe designs that were expected to eliminate flash backs. A study of a number of letters patent indicates quite clearly that there are nearly as many theories concerning the cause of flash backs as there are patented designs. Necessarily considerable ingenuity has been shown in the attempts to accomplish the desired results. Notwithstanding the fact that some of the more recently designed blowpipes lay claim to being safe and nonflash back, still, as will be shown presently, none of those submitted to the tests of this investigation have proved free from flash back under ordinary flash-back test procedure; nor may a great many of them be considered entirely safe pieces of apparatus. The fact is that the theory back of the causes of flash back does not appear to have been fully comprehended and therefore naturally not correctly applied. Neither do any of the treatises or published articles

dealing with oxyacetylene blowpipe design or the autogenous welding process give any statements that indicate that flash-back phenomena have been understood. It is essential then that an explanation of the phenomenon be offered, especially as indicated above, as it is intimately connected with the question of sound welds. The explanation offered is based on some very old principles.

Sir Humphry Davy in the course of the researches that led to the invention of the miner's safety lamp, among other things, showed that there were certain governing principles relative to the inflammability of gaseous mixtures. He pointed out that there was a relation between the richness of the gaseous mixture and the rate of propagation of flame through it, and that such gaseous mixtures could either be enriched to the point where they ceased to be inflammable or that they could be diluted to the point where they would cease to be inflammable. It was also shown that the points where inflammability ceased were very sharply defined, but that the limits could be increased by increased temperature or pressure; and in fact that at these "critical proportion mixtures," at dilution and concentration, a very slight temperature rise will produce an inflammable mixture. It was also shown that the maximum velocity of flame propagation occurred with the gaseous mixture so proportioned that, at the completion of the combustion process, stable compounds were formed. Finally it was shown that if a gaseous mixture is ejected through a tube into the air at a velocity greater than the velocity of flame propagation for that mixture the mixture may be ignited and will burn at the exit opening of the tube without striking back into the interior of the tube. These fundamental principles were later proved to be correct in every particular by Bunsen and since then by a great number of other investigators.

The chemistry of the combustion process of a gaseous acetylene-oxygen mixture shows that the complete oxidization of one volume of acetylene requires two and one-half volumes of oxygen. Experience with the design and operation of oxyacetylene blowpipes has demonstrated the fact, however, that in reality it is only necessary to project, with the blowpipe, equal volumes of oxygen and acetylene, chemistry showing that it is a two-phased process, the second phase of which can be completed by the absorption of the remaining required oxygen from the atmosphere surrounding the flame.

The above shows that the design of the oxyacetylene welding blowpipe is based upon the fact that equal volumes of acetylene

and oxygen are to be mixed and ejected through the tube of the welding blowpipe tip at a velocity at least equal to or in excess of the velocity of flame propagation for that mixture. Under such conditions the flame will burn steadily and freely at the exit end of the tube, gathering enough oxygen from the surrounding air to complete the oxidization of the components of the acetylene gas.

A long experience has seemed to show, nevertheless, that the fulfillment of the above conditions was not sufficient in itself to maintain always the flame at the tube or tip end. It was found that if a blowpipe tip is brought too near the surface of the weld, is accidentally dipped into the molten metal of the weld, has the aperture partly blocked by flying slag, or is badly overheated, the flame will snap back into the tube, causing a so-called back fire, and may even propagate itself still farther backward until it burns at the mixing chamber, thus causing what is known as the flash-back phenomenon. It is due to the fact that this detrimental phenomenon can be so readily produced that so much energy and money have been expended in the attempt to eliminate it.

Outside of the flash back produced through the severe overheating of the head and tip of a blowpipe, it is apparent that all the other causes mentioned above seem to depend for their action upon a partial or complete blockage of the gas flow. It will be readily perceived that any sort of obstruction that checks the volume flow of the gases also increases proportionally the velocity of exit of the gaseous mixture at the tip end. Such being true, it would seem that, with the velocity of exit increased above the velocity of flame propagation, the flame could not strike back into the tube, in other words, back fire. Yet, in spite of this, back fires are caused, and it is known that with some blowpipes a back fire will immediately become a flash back, while in others it causes the extinguishing of the flame, or, if the blowpipe is withdrawn from the work, it will be blown forward again to the exit of the tip or tube. Why should some blowpipes be so much more susceptible to the development of flash back than others?

The lack of sufficient velocity of exit of the gaseous mixture has been accredited as the principal cause for flash back in some of the latest designed and most successful nonflash-back blowpipes. That there is some reason of far deeper significance than this is evident from a study of Tables 4 and 5. It is true that an obstruction to the flow will cause flash backs, but such obstruction is in reality only a contributing cause.

TABLE 4.—Summary of Tests for "Flash Back," Ordinary Conditions, Test 3

[Key: F=flash back. FOO=flash back in oxygen observation tube. FAO=flash back in acetylene observation tube. b=backfire. r=trouble in relighting. v=flame blew away and valve had to be readjusted.]

Torch No.	Tip No.[a]	Pressures used		Action of torch in tests with—			Velocity of exit	Valve wide open
		Oxygen	Acetylene	Cold steel	Fire brick	Wood		
[b](1)	[b](2)	Lbs./in.² [b](3)	Lbs./in.² [b](4)	[b](5)	[b](6)	[b](7)	Ft./sec. [b](8)	[b](9)
1	6	4.00	5.00	b, r		FOO, b, r	371	Oxygen, 2 trials
	6	4.00	5.00	b, v			371	Oxygen, 2 trials
	8	5.75	7.00			v	380	Acetylene
2	8	8.00	8.00	b	v	v	364	Do.
	10	10.00	10.00	b			397	Oxygen
3	[c]9	6.50	8.00	b			479	Do.
	10	7.50	9.00		F	FOO, F.	427	Do.
4	35	12.00	5.00	F	F	F	366	Oxygen, 2 trials
	35	12.00	5.00	F	F	F	366	Oxygen, 2 trials
	36	14.00	6.00	F, r	F	F	335	Acetylene, 3 trials
	36	14.00	6.00	F, r	F	F	335	Acetylene, 3 trials
	36	14.00	6.00	F, r	F	F	335	Acetylene, 3 trials
5	[d]11	11.00	11.00	b			450	Oxygen
	[e]15	15.00	15.00	b			332	Acetylene
6	7	14.00	6.00	b	b	FAO, b.	406	Acetylene, 2 trials
	7	14.00	6.00	b	b	FAO	406	Oxygen, 2 trials
	9	18.00	6.00	F	F	F	375	Acetylene, 3 trials
	9	18.00	6.00	F	F	F	375	Acetylene, 3 trials
	9	18.00	6.00		F	FAO	375	Acetylene, 3 trials
7	[f]7	10.00	10.00	b	b	b	413	Oxygen
	7	12.00	12.00	b		b	462	Do.
8	6	12.00	6.00	b	b	b	495	Acetylene
	[g]7	14.00	8.00	F, b	F	F	558	Oxygen, 5 trials
	7	14.00	8.00	b	b	b	558	Oxygen, 5 trials
	7	14.00	8.00	b	b	b	558	Oxygen, 5 trials
	7	14.00	8.00	b	F		558	Oxygen, 5 trials
	7	14.00	8.00	b			558	Oxygen, 5 trials
9	[h]8	5.00	4.00	b	b	b	427	Acetylene
	9	8.00	8.00	b	b	F	391	Acetylene, 4 trials
	9	8.00	8.00	b		F	391	Acetylene, 4 trials
	9	8.00	8.00	b		b	391	Acetylene, 4 trials
	9	8.00	8.00			b	391	Acetylene, 4 trials
10	7	8.00	6.00	b, v	b, r	F, r	568	Acetylene, 2 trials
	7	8.00	6.00			b	568	Acetylene, 2 trials
	9	14.00	14.00	b		F	582	Oxygen, 2 trials
	9	14.00	14.00			b	582	Oxygen, 2 trials
11	5	6.00	6.00	b	b	b, r	430	Oxygen
	[i]6	15.00	6.00	b	b	b	483	Acetylene
	[j]6	15.00	6.00	b	b	b	483	Do.
12	8	4.50	4.50	b		b	370	Do.
	9	5.50	5.50	b			335	Do.
13	[k]10	21.00	1.00	F	F	F	323	Oxygen
	12	25.00	1.00	F	F	F	343	Oxygen, 2 trials
	12	25.00	1.00	F	F	F	343	Oxygen, 2 trials
14	[l]8	8.00	8.00	b	b	b	353	Acetylene
	[m]12	12.00	12.00	b			485	Do.

[a] Smaller size tips were used for welding ⅛-inch metal and larger size for ¼-inch metal.
[b] Column numbers.
[c] Small explosions denoting attempt at backfire in tests on brick and wood.
[d] Two backfires in 6 trials on cold steel.
[e] Too hot to hold after test on wood.
[f] Head quite hot.
[g] Half of handle too hot to hold after test on wood.
[h] Tendency to flash back not sustained with brick and wood.
[i] Monel metal tip.
[j] Copper tip.
[k] Torch head very hot.
[l] No back flash after brick was warm.
[m] Tendency to backfire with wood and brick.

TABLE 5.—Summary of Tests for Flash Back, Severe Conditions, Test 4

Torch No.	Minutes before flash back		Conditions prevailing after flash back	
	First trial	Second trial	First trial	Second trial
[a] (1)	[a] (2)	[a] (3)	[a] (4)	[a] (5)
1	1.6	1.96	Tip loose	Tip loose
2	.75	1.13		Slight leak at tip
3	.43	.75	Tip leaks in head when hot. No leak when cold	Tip leaks in head when hot. No leak when cold
4	.40	.38		
5	.50	.50		Slight leak in tip
6	.53	.90	Had to be cooled before relighting.	Tip leaked at head
7	.86	.72	Leak in top of tip, tip loose	Flashed back and relit, leaking around tip, which was loose
8	.68	.65	Tip leaked at joint	
9	.80	.68		
10	.33	.42		Small leak in bottom section of tip
11	.80	.92	Monel metal tip	
11	.38	.58	Copper tip, tip leaked around end	Copper tip, tip leaked aroun d end
12	.87	.68		
13	1.07	.85		
14	.78	.90		Handle too hot to hold

[a] Numbers in parentheses are column numbers.

Partial obstruction of the tip opening by slag, etc., or the confinement of the expanding products of combustion through bringing the tip end too near to the metal surface causes the development of a back pressure. This back pressure may be readily observed by watching a pressure gage as a lighted blowpipe tip is alternately brought in contact with or removed from a plane surface.

This back pressure has a very detrimental effect in that it chokes off the flow of the gas that is admitted under the lower pressure. Reference to Figs. 52 to 59, exhibiting results of tests in which the amount of obstruction of the exit opening was under control, shows the development of this back pressure and the restriction of the flow of the gas admitted under lower pressure.

If then the blowpipe design is such that the back pressure tends to check the oxygen supply more than the acetylene an increased richness (in acetylene) of the gaseous mixture occurs. The gaseous mixture in the blowpipe will therefore have a lower flame propagation velocity due to the fact that the increased richness is tending toward the critical proportion mixture at concentration as defined by Davy. The velocity of flame propagation is therefore decreasing while the velocity of exit has been increased by the partial closure of the orifice. There is a resulting tendency therefore for the flame to be maintained at the tip end. If under the same conditions the obstruction is sufficiently severe, the gaseous mixture within the blowpipe passes the critical proportion mix-

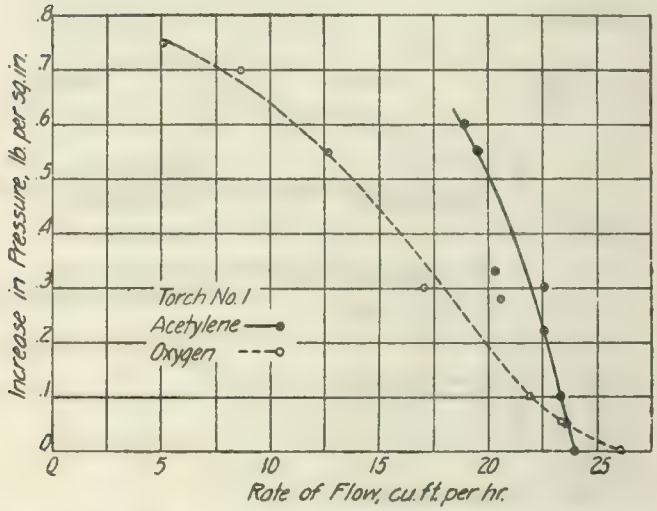


FIG. 52.—*Reduction of flow due to back pressure, torch No. 1. Test run with flame extinguished without manipulation of valves*

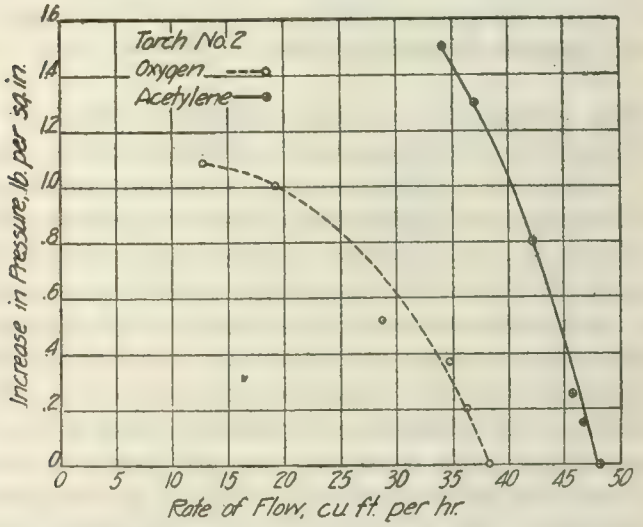


FIG. 53.—*Reduction of flow due to back pressure, torch No. 2. Test run with flame extinguished without manipulation of valves*

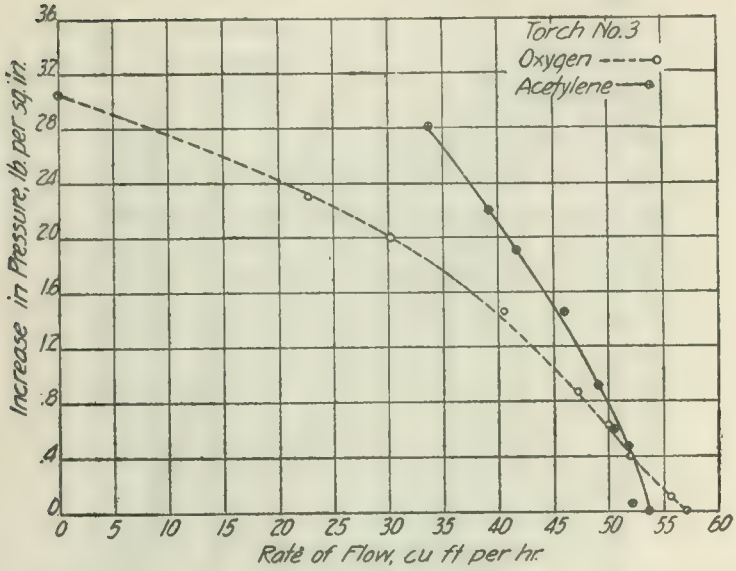


FIG. 54.—*Reduction of flow due to back pressure, torch No. 3. Test run with flame extinguished without manipulation of valves*

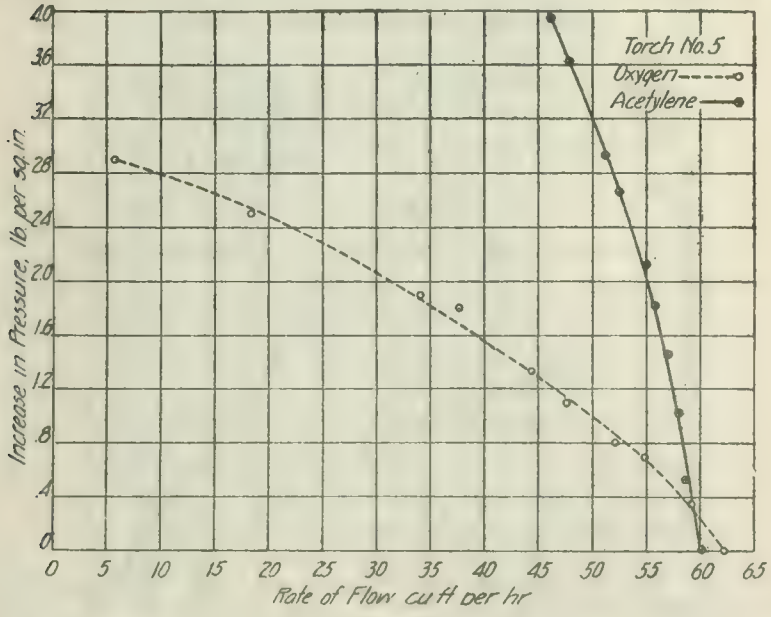


FIG. 55.—*Reduction of flow due to back pressure, torch No. 5. Test run with flame extinguished without manipulation of valves*

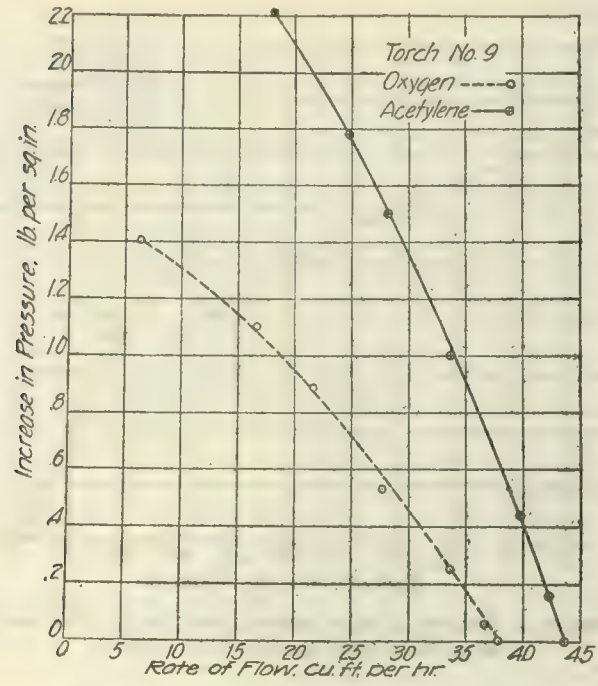


FIG. 56.—*Reduction of flow due to back pressure, torch No. 9. Test run with flame extinguished without manipulation of valves*

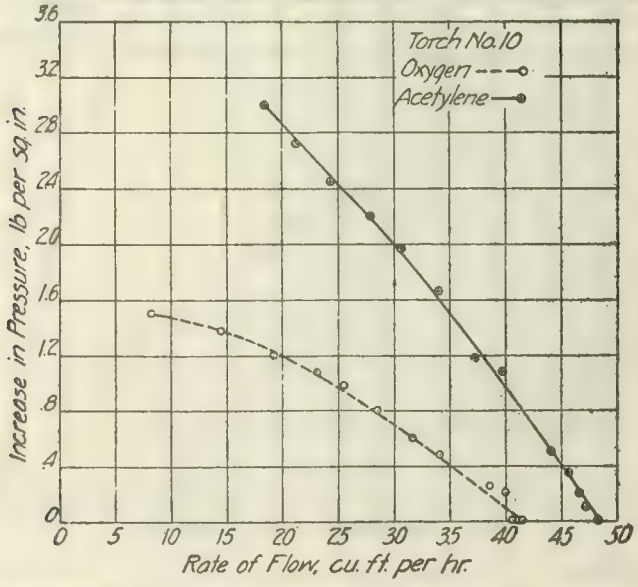


FIG. 57.—*Reduction of flow due to back pressure, torch No. 10. Test run with flame extinguished without manipulation of valves*

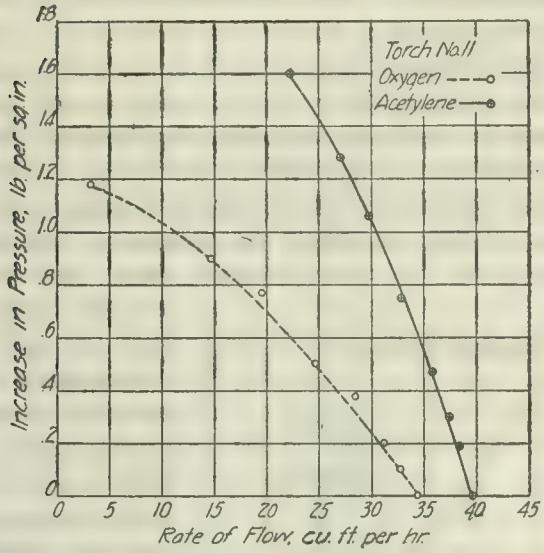


FIG. 58.—*Reduction of flow due to back pressure, torch No. 11. Test run with flame extinguished without manipulation of valves*

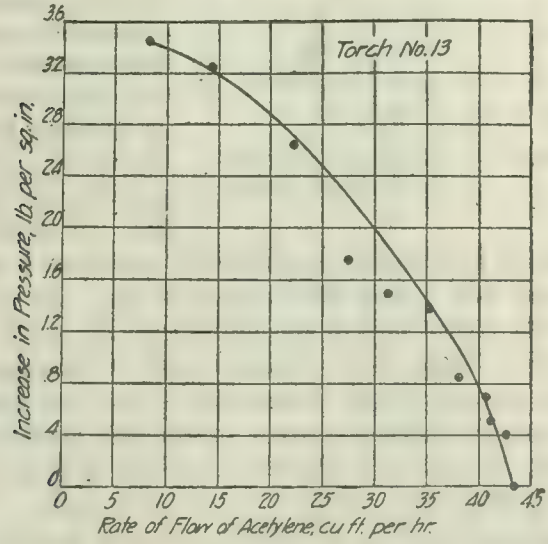


FIG. 59.—*Reduction of flow due to back pressure, torch No. 13. Test run with flame extinguished without manipulation of valves*

ture at concentration, the mixture becomes noninflammable and the flame is extinguished. With blowpipes furnishing the acetylene under a higher pressure it is therefore evident that the tendency to produce flash backs is minimized if not entirely suppressed.

On the other hand, if the blowpipe design is such that the back pressure tends to check the acetylene supply more than the oxygen, a mixture leaner in acetylene is produced. Such a mixture is approaching the two and a half volume of oxygen to one volume of acetylene mixture at which the maximum velocity of flame propagation exists. A back pressure, then, that increases the oxygen content above that of the one-to-one volume ratio furnished in blowpipe design is producing a gaseous mixture that possesses a higher flame propagation velocity than that possessed by the one-to-one mixture.

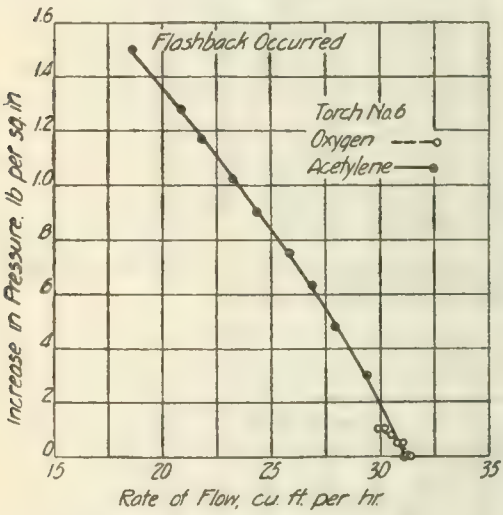


FIG. 60.—*Reduction of flow due to back pressure, torch No. 6. Test run without extinguishing flame or manipulating valves*

If the increased exit velocity caused by the obstruction to the gas flow compares favorably with the increased velocity of flame propagation, the ignited mixture will continue to burn at the tip. In all blowpipes of this type which have been examined this condition is not fulfilled, with the result that a flash back always occurs. As experimental evidence of the correctness of this statement the data of Fig. 60 are presented. These were obtained with a blowpipe mounted so that the amount of the obstruction to gas flow could be controlled. When the oxygen content of the gas mixture had been increased to the amount indicated by the diagram the flame snapped back of its own accord to a true flash back.

Having snapped back to the mixing chamber the back pressure will be destroyed and the flow becoming normal again, the flame burns there in the volumes permitted by the blowpipe construction with insufficient oxygen to cause complete oxidization and the acetylene gas is broken up into its components one of which

unites with the available oxygen to form water vapor, leaving carbon deposited on the walls of the tip tube and mixing chamber. This explains not only the heavy carbon deposit that is always left as the result of a flash back but the reason why it is relatively so moist.

It is of course possible that if the oxygen content of the gaseous mixture could be increased beyond the proportions for maximum velocity of flame propagation, that is, beyond the two and a half to one volume ratio, the velocity of flame propagation might be reduced sufficiently to compare favorably with the exit velocity. In the light of the above, however, it is evident that this condition can not be produced, nor would it be desirable, as the flame would be extremely oxidizing.

At how great a disadvantage the high pressure oxygen blowpipe designs are as compared with those operating with acetylene delivered at excess pressure is very apparent from another consideration. Eitner [4] has shown that the lower inflammable limit by percentage volume of combustible gas in a mixture of acetylene and air is about 3.35 per cent, and that the upper limit is about 52.3 per cent. There is no doubt that with a more active gas, such as oxygen, the lower limit will be at least the same and possibly the upper limit will be found to be slightly raised. Bearing in mind that normally a 50 per cent mixture is being passed to the mixing chamber, it is readily seen that the critical proportion mixture at concentration is reached very soon, while the critical proportion mixture at dilution is not reached until almost the entire combustible gas supply has been shut off.

It has been known for some time that an operator could extinguish a flash back in a high-pressure blowpipe by suddenly increasing the oxygen pressure or by crimping the acetylene supply line. The reasons are now very apparent. In the former case the oxygen volume ratio is greatly increased by the accelerated checking of the acetylene supply until the volume ratio is diluted beyond the maximum explosive mixture, the flame propagation velocity is thereby reduced, and at the same time the increased pressure of the oxygen expels the mixture at increased velocity. The flame may therefore be pushed to the outlet opening of the tip. In the latter case, crimping the hose line, the combustible gas is shut off and the flame may be pushed forward as above or may go out from the gas mixture having reached the critical proportion

[4] L. A. Groth, Welding and Cutting Materials by Aid of Gases or Electricity, Archibald Constable & Co. (Ltd.), London, p. 251; 1909. See also G. A. Burrell and G. G. Oberfell, the Explosibility of Acetylene, Bureau of Mines Technologic Paper No. 112, p. 5; 1915.

mixture at dilution; but in this case, on account of the low percentage mixture at which this occurs, it may as well be caused through the entire combustible gas supply having been cut off.

Having explained the conditions necessary for, and the action of flash-back phenomena, it is of interest to determine why some blowpipes flash back so much more readily than others of similar design; as, for example, those in which the oxygen pressure is higher than that of the acetylene. The rapidity of development of the condition for producing a flash back depends entirely upon the rapidity with which the back pressure is set up. That there is a difference in the rate at which the back pressure is developed is evident from an inspection of Figs. 52 to 59. The data of these tests show that those blowpipes which were more susceptible to flash back from obstruction of gas flow were those in which the back pressure most rapidly diminishes the acetylene flow. A blowpipe operating on the injector principle or one in which the oxygen pressure is very much in excess of the acetylene pressure will therefore be far more susceptible to flash-back phenomena.

An investigation of the actual pressures specified by the manufacturers, as well as a study of certain letters patent, indicates that a few of them at least, in their experimental work, have become cognizant of the effect a gas mixture rare in acetylene has upon the development of a flash back. To overcome flash-back troubles, several of the recent designs are based upon furnishing acetylene under a higher pressure, spoken of in the letters patent as higher velocity, than that used for the oxygen. Reference to Tables 4 and 5 shows that flash backs can be caused in apparatus of that type. The question may rightly be asked as to why flash backs can then be developed in such blowpipes.

A critical study of the design of the blowpipe tips shows further that the way in which one stream of gas is admitted into another has a decided effect upon conditions resulting from the development of a back pressure.

A back pressure caused by tube obstruction pushes back into the tube in the most direct line (Fig. 61). An acetylene gas passage entering the oxygen tube at right angles has its gas flow cut off very quickly, especially when the oxygen pressure is much higher than the acetylene pressure and the point of admission of the oxygen is, as it is in most of the tip designs, beyond the point of admission of the acetylene.

In this respect a gas passage entering the oxygen passage at a sharper angle is more advantageous, but, under certain conditions, this construction may not produce results that are any

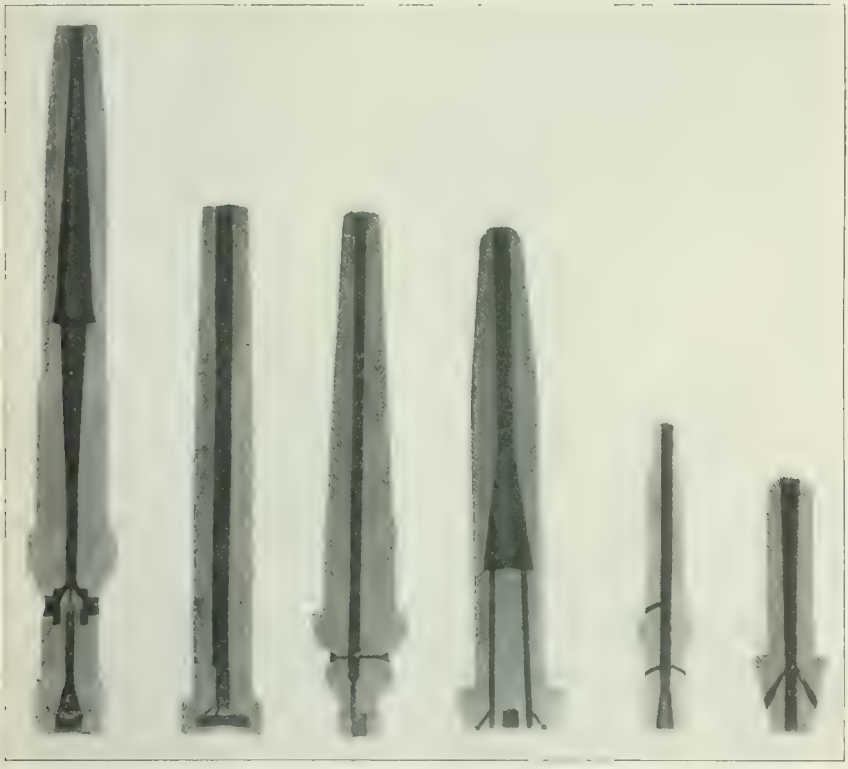


FIG. 61.—*Views of cross sections of tips*

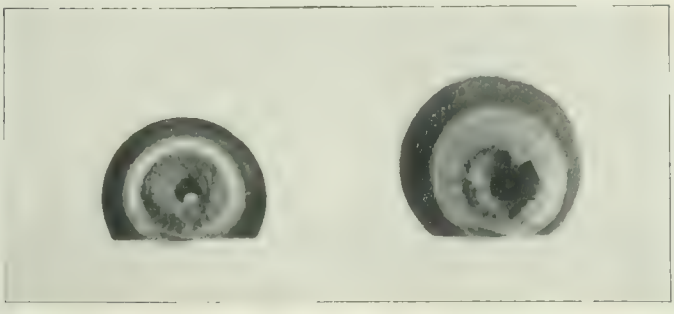


FIG. 62. *Tips after a flash back*

more desirable. If the velocity of the oxygen jet is relatively high and the acetylene gas ports are placed at an angle to the tube of the oxygen jet, an aspirator or injector action is developed relative to the acetylene. The higher the velocity of the oxygen the greater is the aspirating effect on the gas tube. This aspirating effect is in reality an attempt to produce a partial vacuum within the acetylene tube. If, then, while operating under such conditions a sudden obstruction of the exit passage occurs with its accompanying back pressure development, the sudden release of the aspirating action causes a collapse of the partial vacuum and the gas passage tube is therefore momentarily choked or at least throttled by the infiltration of oxygen. This action instantly produces the leaner mixture and conditions necessary for the development of a flash back.

But there are also two other fundamental physical laws which apparently have not been given proper consideration in the designs of blowpipes, and which will probably account in a far larger measure for the development of flash-back conditions. These laws refer to:

1. The variation of velocity of a fluid in a pipe of varying size.
2. The diminution of pressure in a throat.

The first law recognizes the fact that the mass discharge of a fluid at all sections is a constant; hence for a pipe of varying size, as shown, the following condition holds when the friction loss of pressure is neglected:

If $a > a'$, then $av < a'v'$, where

a and a' = areas of gas passage,

v and v' = velocities of gas.

This indicates that at various points along a tube the velocities of the fluid increase as the section areas decrease.

The second law is based on the fact that, neglecting friction and eddy losses, at any two sections of such a tube the total energy per unit mass will be equal; that is, an increased velocity produces a decreased pressure.[5]

---

[5] The relation for a gas is given by the equation

$$v'^2 - v^2 = 2TC_p\left[1 - \left[\frac{p'}{p}\right]^{\frac{C_p - C_v}{C_v}}\right], \text{ in which}$$

p and p' = pressure;
v and v' = velocities;
d = density;
T = absolute temperature;
C_p = specific heat at constant pressure;
C_v = specific heat at constant volume.

The decrease of pressure with increase of velocity is perhaps more clearly shown by the approximate formula

$$p + \tfrac{1}{2}dv^2 = p' + \tfrac{1}{2}dv'^2,$$

which neglects the expansion of the gas.

From these two laws it is evident that if the area of a tube is contracted with the idea of securing increased velocity the pressure in the contracted area is reduced.

Reference to Fig. 61 shows that in practically all designs the contracted throat is used in the acetylene passages. It will be readily realized then that in even those blowpipes which are supposed to operate with acetylene delivered at excess pressure there exists in the fundamentals of their design an inherent tendency for the choking off of the acetylene supply and the development of the leaner flash-back mixture at the first indication of back pressure.

The inherent defects in the present designs show plainly then why it is also possible to produce a flash back almost at will in the blowpipes that are designed on the excess acetylene pressure basis. It is evident, therefore, that none of the blowpipes designed for use with one gas delivered at a higher pressure than that used for the other are really free from flash-back troubles caused by obstructions to the gas flow.

There is another important consideration to be discussed relative to blowpipes using one gas at higher pressure, and that is their safety during operation. The foregoing discussion shows that an obstruction to the gas flow issuing from the tip sets up a decided and effective back pressure and that the back pressure thus set up will be sufficient not only to check the flow of the gas admitted under the lower pressure, but also actually to push it back into its supply tube and line. The return of one gas into the supply line of the other instantly forms an explosive gas mixture back of the mixing chamber. Reference to the exceedingly low limit of the critical mixture at dilution as indicated by Eitner's investigations shows how rapidly a dangerous mixture can be formed by a blowpipe using acetylene under excess pressure. A blowpipe using the oxygen under the excess pressure is about equally as dangerous. It is simply a question of the time interval the back pressure has available to accomplish its results.

The degree of danger from this cause is greater, the greater the difference in pressure. A blowpipe operating with one gas at a pressure greatly in excess of the other can force a return mixture just so much more quickly than a blowpipe operating with less difference in pressures. Also with one pressure greatly in excess of another a larger amount of explosive mixture will be formed back in the tube or line, as the distance the explosive mixture will be forced back depends upon how quickly the equalizing

element, the pressure developed by the compression of the gas in the tube, overcomes the effect of the back pressure.

In connection with this phenomenon of the creation of explosive mixtures in tubes and lines due to back pressure, it seems desirable at this time to call attention to the decidedly dangerous influence leaks in the supply lines can have in cases where a dangerous mixture is being forced backward in one tube.

With the knowledge that the back pressure set up by the obstruction of gas flow from a tip can return a dangerous explosive mixture into the tubes and supply lines, and further with the realization, now apparent at least, that none of the unequal pressure blowpipes are free from the possibility of the development of a flash back, it can be readily seen that no blowpipe on the market to-day that operates under unequal or unbalanced pressures can with assurity be declared absolutely safe. This is especially true as it has not as yet been established how small an opening will have to be before it will cease to propagate a flame through an explosive mixture of oxygen and acetylene. It is known, though, from the investigation carried on to determine the feasibility of acetylene lamps for mine use, that the opening is very small. That the return of such explosive mixtures can be caused is evidenced by reference to Table 4.

Recognizing the dangers coincident with blowpipes operated under dissimilar pressures, some manufacturers have attempted to minimize the return of dangerous gas mixtures by incorporating check valves in the blowpipe construction. While such desires are commendable they are in their method of fulfilment of doubtful value in that there is no assurance that the valves will always be operative. Failure, or loss of life in springs, or pieces of dirt or scale, or corrosion of a stem or seat may readily cause a check valve to fail to function. Its utility then is very questionable.

Other manufacturers have sought safety in action by attempting a design based on delivering the gases under equal pressure. Possibly also they have realized to some extent the effects of excess oxygen or acetylene on the development of flash-back phenomena. A very careful study of their letters patent shows, however, that they have not completely understood the phenomenon, its cause and effects, and the methods necessary for its suppression. It is therefore not surprising that not one of the so-called equal pressure blowpipes proves actually to be such. The data of Table 4 show this beyond a doubt. They are therefore at times subject to the same flash-back difficulties and the same lack of safety in

operation, their differences being only a question of relative degree and subject to the same principles mentioned above.

In mentioning above the things that seemed to be responsible for the flash back, specific reference was made to overheating of the tip and head. All blowpipes can be made to flash back from such a cause. How quickly they will do so seems to depend entirely on their design. It is certain that water-cooled heads can assist in restraining such flash backs. It is also apparent that a tip and head made of metal of high thermal conductivity, pure copper, for example, is beneficial. In fact, any construction that tends to maintain a cool gas flow seems to be helpful, and it is probable that in that fact lies the explanation for flash backs caused by overheating of blowpipe parts.

It will be recalled that Davy's experiments showed that the critical mixture proportions were affected by changes in temperature, that a slight increase in temperature may make what is normally a noninflammable mixture inflammable. It is known also that the velocity of propagation of flame is increased with increases in temperature. It is evident, then, that under the influence of a high temperature there is an increasing tendency for even a blowpipe designed for operation with excess acetylene to flash back on the development of back pressure.

The real cause for flash backs developed under the influence of high temperature, where back pressure does not necessarily exist, is probably due to the sudden heating to the ignition point of a large volume of gaseous mixture. The tip and head become extremely hot for their entire length and subject a relatively large volume of gas mixture to the effects of the high temperature. The result is naturally a detonation. The suddenness and violence with which such flash backs occur and their destructive effects on the tips ends, as shown in Figs. 62 and 63, are evidence that the burning gaseous mixture has been subjected in reality to a detonation.

The difference in action that takes place in the two forms of flash back, namely, those caused by back pressures and those caused by detonation of the gaseous mixture, can probably be best illustrated by calling to mind the familiar facts concerning gun cotton, a material that, lying loosely and open to the air, will burn harmlessly if ignited by flame, becomes a most violent explosive if fired by a percussive detonator. It is a question of the rapidity with which a large volume of the mixture is ignited.

FIG. 63.—*Tips after a flash back*

FIG. 64. *Oxyacetylene flames*

A Acetylene flame, no oxygen
B. Slight excess of acetylene
C. Neutral flame

If, then, the sudden firing of a relatively large volume of a gaseous mixture produces a detonation, it is quite evident that the rate of flame propagation is exceedingly high. For an oxygen-hydrogen mixture it has been reported as approximately 9315 feet per second; for a carbonic-oxide mixture, a relatively slow and inert combination, at about 3575 feet per second.[6] Figures for acetylene do not seem to be available, but it is certain that they will be nearly as high as those for hydrogen.

The futility of attempting to prevent flash backs from this cause by ejecting the mixture at a high velocity is very apparent Knowledge, however, of the cause of the action indicates that anything that tends to keep the gaseous mixture cool tends to hinder the development of such flash backs.

Having discovered the causes of the development of the so-called flash-back phenomenon, and shown that the safety of a blowpipe is dependent upon the causes back of this same phenomenon, it is next in order to inquire as to what possible effect the causes of flash backs may have upon the gas ratio; that is, upon the economy of operation and upon the process of autogenous welding.

The questions are hardly proposed before their answers are apparent. It cannot be expected that a blowpipe whose gaseous volume mixture is constantly changing from the one-to-one volume ratio to an excess carbon (acetylene) content or an excess oxygen content can maintain a constant gas ratio or a truly neutral flame. If this is so, it is hopeless to expect such blowpipes to be capable of producing unoxidized or uncarbonized welds.

The controversy on the causes for unsound welds has of course been two sided. It has been said that even experienced welders excuse the shortcomings of their work by claiming that the blowpipe or the welding rod was not good. But generally up to this time the condemnation has been placed principally upon the welder and there is no doubt in certain cases rightly so.

In certain quarters the oxyacetylene weld has been severely criticized. As a result of this criticism it has been suggested that all welders should receive special training, that the quality of their work be constantly checked by tests, and that they be licensed. Extensive metallurgical studies of welds, and attempts to find nonoxidizing welding rods have also been made. All these are good and useful expedients, but if in the first place the operator is not furnished with a blowpipe that has been designed

[6] D. Clerk, The Gas, Petrol, and Oil Engine, p. 116.

with a correct interpretation and application of fundamental physical laws, it can not be expected that sound welds will be forthcoming, no matter what other expedients are employed to increase the efficiency of the weld.

The oscillatory motion given to the blowpipe in the process of welding and the trembling of the operator's hand, by causing a constantly varying back pressure as the blowpipe tip approaches or recedes from the surface of the weld insures a constantly fluctuating gaseous mixture with a predominating tendency, as indicated above, toward an oxidizing flame.

In the more careful welding operations very great pains are usually taken to burn out oxide or slag pockets, etc., with the result that the tip end is forced down into a confined space. Even with such care welds are found to be oxidized. The bottom of the V of a weld is almost always oxidized, while the upper portion may show more or less of the clean cast metal. It has been known for years that a shaky-handed welder could not make sound welds. It has also been known that certain blowpipes, as regards flash back, were particularly susceptible to the nearness with which the blowpipe tip approached the welding surface. Further, some at least have known that after placing an ignited and properly adjusted blowpipe flat on a surface and then gradually raising the head, keeping the tip upon the surface meanwhile, the blowpipe would flash back after the head had been elevated to a certain angle. These are some of the facts that indicated that there was being developed at such times the detrimental back pressures.

All the data available as a result of the investigation bear out the logical conclusions of the above theory—the high gas ratios during welding, the difference in gas ratios when identical tests are performed by different operators, the lower gas ratios when a blowpipe burns freely in air (see Table 6), the fluctuations in volume flows as determined by flow-meter readings taken every few minutes with gas pressures maintained within a few hundredths of a pound per square inch throughout the entire test period. (See Fig. 21.) Most convincing of all is the wide variation in the physical properties of the welds made during the investigation, a group of welds that probably was made with greater care than has ever been bestowed upon any other such set. (See Figs. 65 to 71.)

All past experience, including the experience gained during these tests, points to but one convincing conclusion, namely, that a blowpipe designed to be absolutely free from flash back

caused by any form of obstruction, under all working conditions, will also be the eminently safe blowpipe and the blowpipe that will with ordinary care produce sound welds. Such a blowpipe will be one so designed that under all conditions of operation, even to complete blocking of the gas exit at the tip end, there will be maintained a one-to-one volume delivery of each gas at identical pressures. No oxyacetylene blowpipe on the market to-day fulfills these fundamental conditions.

A few other points are of interest in connection with the welding blowpipe investigation. It was mentioned above that one of the most noticeable outstanding features which was the first to be investigated in the attempt to solve the apparent discrepancies in welding blowpipe data was the high gas ratio obtained during welding operations. Attention has also been called to the fact that such ratios were lower when the blowpipe burned freely in air. The reason for this is of course now apparent. But it will be of interest to note that even such values were relatively high, and this notwithstanding the further fact that in a majority of gas-ratio tests the neutral flame was set in the presence of and met the approval of experienced blowpipe operators, representatives of the manufacturers. Consideration of these facts led to the belief that what is called the neutral flame is more or less masked by the operator's personal equation, that is, what he considers a neutral flame to be. Further, continued experience indicated that when the gases were issuing at a high velocity the extreme ragged edge of the inner cone seemed to interfere with the determination of the neutral flame. It became apparent, then, that the general tendency was toward setting an oxidizing flame. This was strengthened by the knowledge that all treatises and books of instruction that were available describe the neutral flame as having been secured when the intensely white flame or inner cone is "sharp in outline—symmetrical and smooth." (Fig. 64.) The trouble with this statement lies in the emphasis given to the words "sharp" and "smooth." In adjusting the burning gases to produce a neutral flame it seems to have become an ingrained action upon the part of the operator to cut the acetylene gas flow back until the entire inner cone is "sharp" and "smooth," with the result, as exhibited by the gas-ratio tests of this investigation, that the flame has actually been made quite oxidizing.

Having come to this conclusion, the idea of examining the flame spectroscopically presented itself, and for this purpose,

because of its convenience, a replica diffraction grating was used. The use of this method suggested itself, as it was recognized that in a carbonizing flame there must exist unburned incandescent solid carbon particles which would superimpose their blurred continuous spectra over the line spectra of the flame. Such proved to be the case, and it was found, as anticipated, that probably too much attention has been paid in the past to setting the inner cone to too sharp a contour, especially on the extreme end. The results of a series of gas-ratio tests run with the use of the gratings are shown also in Table 6, and it will be noticed that they show very marked changes in values and much greater uniformity.

This table shows that practically any of the present blowpipes can be made to produce a neutral flame and burn equal volumes of oxygen and acetylene if the flame can burn undisturbed in the air. But, as stated above, none of them can maintain such a flame during the welding process.

TABLE 6.—Summary of Tests for Gas Ratios

Torch No.	Gas ratios for test No.—						
	1–a	1–b	2	5–a–1	5–b	With gratings	
						5–a–1	5–b
[a] (1)	[a] (2)	[a] (3)	[a] (4)	[a] (5)	[a] (6)	[a] (7)	[a] (8)
1	1.13	1.16	1.14	1.19	1.08	1.04	1.01
	[b] (1.11)	(1.15)	(1.14)	(1.18)	(1.04)	(1.05)	(.981)
2	1.12	1.07	1.08	1.04	1.06	1.04	.992
	(1.11)	(1.06)	(1.06)	(1.04)	(1.04)	(1.02)	(.996)
3	1.21	1.26	1.13	1.10	1.07	1.01	1.04
	(1.18)	(1.23)	(1.19)	(1.07)	(1.09)	(1.01)	(1.09)
4	1.13	1.09	1.41	1.14	1.29	.....	.....
	(1.13)	(1.09)	(1.16)	(1.15)	(1.10)	.....	.....
5	1.07	1.05	1.03	1.02	1.04	1.01	.999
	(1.05)	(1.04)	(1.13)	(1.02)	(1.10)	(1.01)	(1.03)
6	1.10	1.11	1.14	1.12	1.17	1.06	1.08
	(1.10)	(1.11)	(1.15)	(1.12)	(1.16)	(1.03)	(1.07)
7	1.07	1.05	1.03	1.04	1.07	1.05	1.02
	(1.04)	(1.03)	(1.02)	(1.01)	(1.02)	(1.01)	(.994)
8	1.13	1.12	1.18	1.27	1.43	1.05	1.01
	(1.12)	(1.12)	(1.16)	(1.28)	(1.38)	(1.03)	(.986)
9	1.15	1.18	1.19	1.14	1.14	1.43	1.19
	(1.18)	(1.24)	(1.17)	(1.14)	(1.15)	(1.39)	(1.15)
10	1.12	1.15	1.19	1.10	1.04	1.03	1.01
	(1.11)	(1.13)	(1.06)	(1.09)	(1.05)	(.989)	(.992)
11	1.19	1.20	1.26	1.27	1.09	1.21	1.04
	(1.19)	(1.21)	(1.23)	(1.28)	(1.08)	(1.19)	(1.05)
12	1.21	1.21	1.13	1.02	1.04	1.02	1.02
	(1.17)	(1.20)	(1.11)	(1.01)	(1.02)	(.998)	(.979)
13	1.02	1.13	1.09	1.09	1.07	1.02	1.00
	(1.04)	(1.12)	(1.09)	(1.08)	(1.06)	(1.01)	(1.00)
14	1.07	1.09	1.08	1.08	1.10	.999	.956
	(1.06)	(1.08)	(1.07)	(1.08)	(1.10)	(.992)	(.964)

[a] Column numbers.
[b] Values in parentheses are computed from flow-meter data.

TABLE 7.—Summary of Results of Tensile and Bend Tests of the Oxyacetylene Welds Made in Tests 1–a, 1–b, 1–c, 1–d, and 2

Torch No.	Ultimate tensile strength and angle of bend, test No.—									
	1–a		1–b		1–c		1–d		2	
	U. t. s.[a]	Inc. ∠[b]	U. t. s.	Inc. ∠	U. t. s.	Inc. ∠	U. t. s.	Inc. ∠	U. t. s.	Inc. ∠
	c(2)	c(3)	c(4)	c(5)	c(6)	c(7)	c(8)	c(9)	c(10)	c(11)
1	39.3	116	44.4	115	33.1	44	42.1	77	43.6	51
	44.9	121	47.4	95						
2	41.2	29	39.4	39	41.2	67	45.8	58	43.7	69
	44.1	56	43.9	67						
3	31.9	50	27.9	28	[d] 43.0	[d] 94	35.3	37	33.2	30
	32.1	36	35.9	20	[e] 38.9	[e] 62				
4	36.2	62	41.4	95	40.6	61	39.0	107	53.5	117
	43.0	41	36.7	73						
5	43.0	72	31.1	63	43.6	34	32.1	18	33.0	52
	46.3	25	37.7	63	[e] 42.9	[e] 42				
6	40.6	72	45.8	59	34.4	16	44.3	56	48.6	44
	43.7	69	42.3	65						
7	44.2	33	37.3	56	36.1	73	36.3	53	48.2	49
	46.0	77	40.8	65						
8	43.4	43	39.8	64	32.9	20	39.5	80	41.4	74
	43.8	70	42.6	60						
9			34.9	38	42.0	67	44.1	71	44.9	57
	46.5	62	45.2	39						
10	41.9	54	35.1	26	[d] 47.7	[d] 89	45.6	72	39.4	23
	43.1	59	39.6	51						
11	43.8	54	38.7	66	37.9	36	38.4	96	45.1	39
	27.0	51	29.2	57	32.8	38				
12	40.4	43	39.0	54	42.3	60	49.4	94	45.8	62
	40.2	32	39.5	56						
13	39.3	51	31.0	83	40.1	61	34.4	82	48.3	58
	52.1	61	41.6	24						
14	40.4	63	45.6	107	39.4	52	42.4	65	38.1	38
	41.0	49	46.5	65						
Average: First plate	40.4	57	38.0	64	38.6	49	40.6	69	43.3	54
Second plate	42.4	58	40.6	57	32.8	38				
General average[f]	41.4	57	39.3	60	35.7	44	40.6	69	43.3	54

Average ultimate tensile strength of all welded ½-inch plates..... 39.2 lbs./in.²
Average ultimate tensile strength of unwelded ½-inch plates..... 54.9 lbs./in.²
Efficiency of welds in ½-inch plate..... 71.4 per cent.
Tests numbered 1 were welds made with ½-inch plates; test 2 was made with ¾-inch plate.

[a] U. t. s.=ultimate tensile strength in thousands of pounds per square inch.
[b] Inc. ∠=included angle of bend, cold-bend test. Bottom of V in compression. Pin diameter equal to thickness of metal. Included angle for unwelded plate=180°.
[c] Numbers in parentheses are column numbers.
[d] Weld made at −50 per cent pressure instead of +50 per cent pressure.
[e] Weld made at normal pressure instead of +50 per cent pressure.
[f] Welds referred to in footnotes [d] and [e] are not included in the general average.

Referring again to the strengths of the welds executed during these tests (see Table 7), it will be seen that the second plate welded during each test invariably showed higher strengths. It will be remembered that to counteract the effects of expansion the 2-foot test welds were made by welding two 1-foot lengths of plate. These plates were supported on a heavy iron casting that contained a channel throughout its length parallel to and directly under the line of welds. This channel caused the flame of the blowpipe to

return under the plates being welded and thus preheated to some extent the second plate. Further, it caused a decided heating of the near end of the base casting. The explanation for the higher strength shown by the second plates probably lies in the more uniform preheating of the second plate and the greater annealing effect produced by the heated base casting, the latter causing a release of the tensile strains resulting from the contraction of the metal along the line of the weld.

The average strengths and average included angle of bend are given in the above-mentioned table (Table 7) for what they are worth. It is very probable in the light of present knowledge of the requirements of blowpipe design, that some new ideas will be forthcoming concerning the average strength of oxyacetylene welds. Finally in Figs. 65 to 71 are exhibited microphotographs showing the effect upon the grain structure of the material, in this case mild steel, of the autogenous welding process, effects not necessarily detrimental when properly performed, but, as exhibited by the microphotographs, instructive.

VII. SUMMARY

In conclusion, the results of this investigation would seem to warrant the following statements.

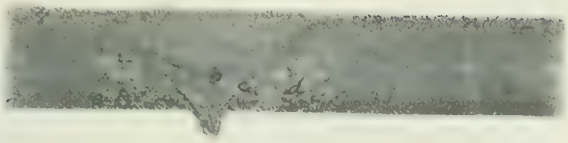
For the Cutting Blowpipes.—1. There is to-day no generally accepted theory for proportioning, for the cutting of metal of various thicknesses, the volume and velocity of the issuing cutting jet, with the result that none of the apparatus submitted to test proved economical for all thicknesses.

2. There is for any thickness of metal cut a limiting velocity of exit of the cutting jet, at which complete utilization of the oxygen takes place and a limiting value for the amount of oxygen required to produce a cut.

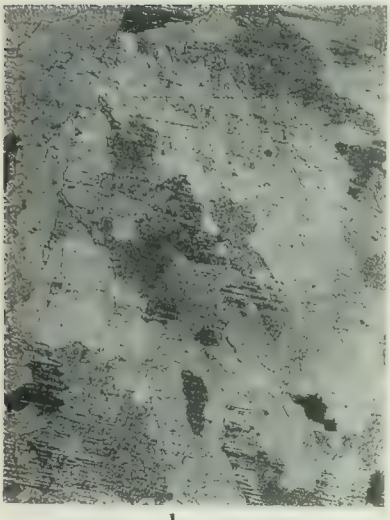
3. That an increase in acetylene consumption, in oxygen consumption, or in the velocity of exit of the cutting jet beyond the limiting values does not produce increased efficiency in commensurate ratio.

4. That a large majority of the blowpipes tested were equipped with excessive preheating flames for the thickness of metal the tip is specified for, and that such excessive sized flames are disadvantageous both from the standpoint of economy of operation and quality of work performed.

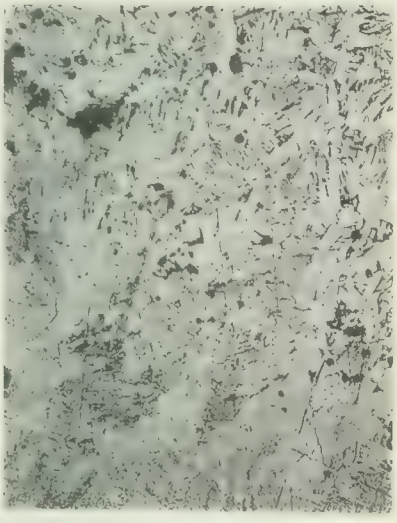
5. That considerable improvement in economy of operation seems possible in cutting material of 2-inch thickness, and that



a



b



c

d

e

FIG. 65.—*Microstructure of weld No. 1*

a. Welded plate, natural size. [illegible]
Weld metal
Junction of weld and plate
Overheated plate metal near weld
c. Unaltered plate metal

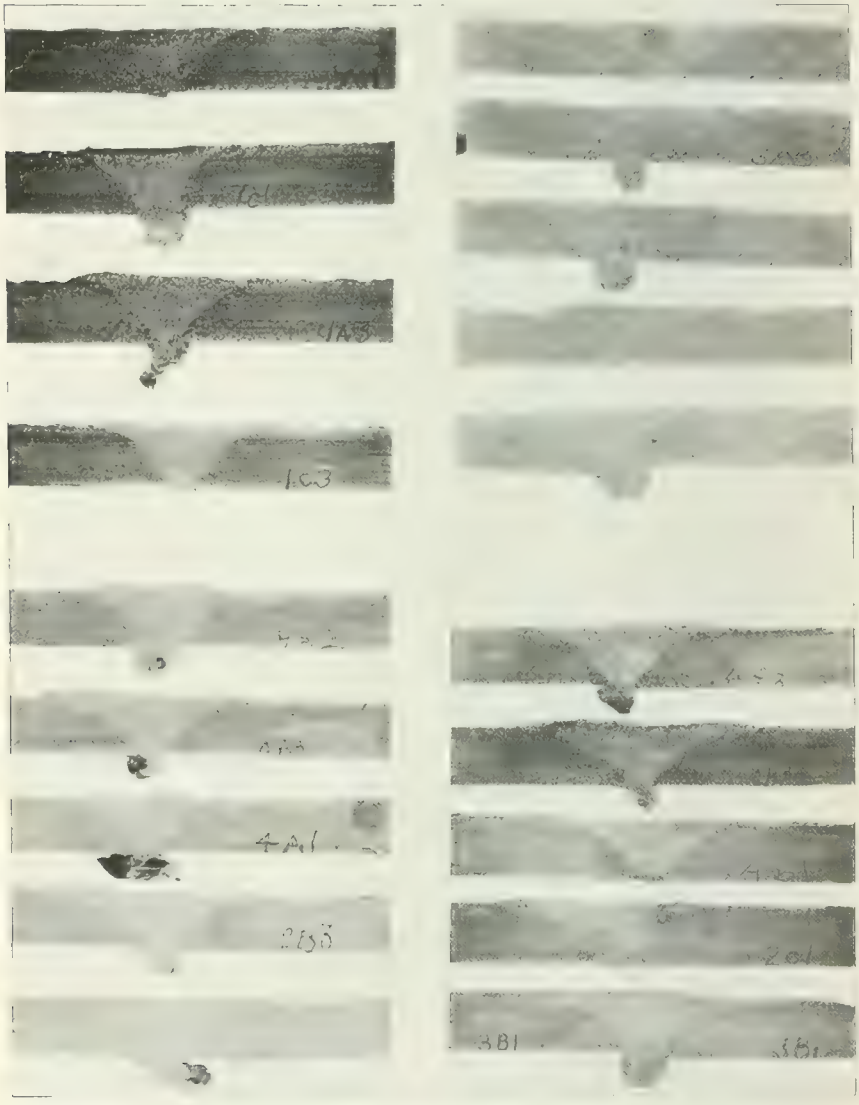


FIG. 60.—*Etched sections through welds*

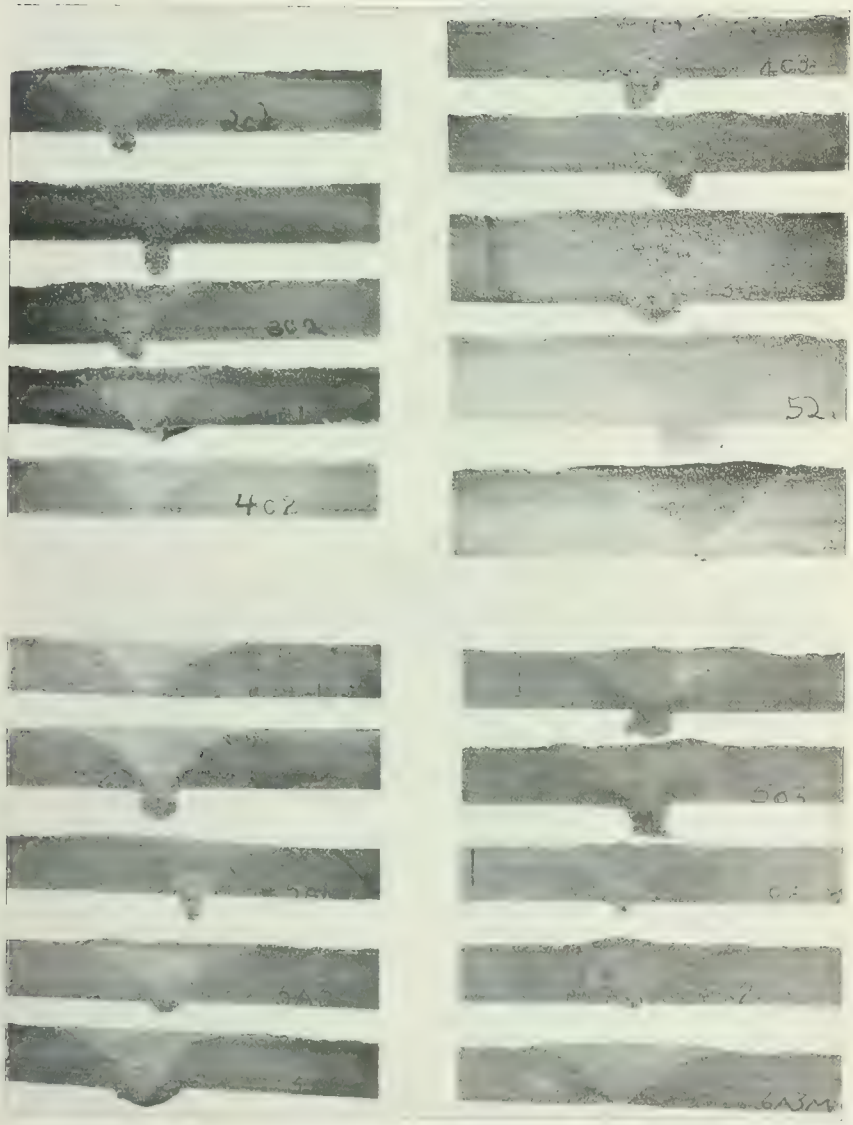


FIG. 17.—Etched specimens through welds—Continued

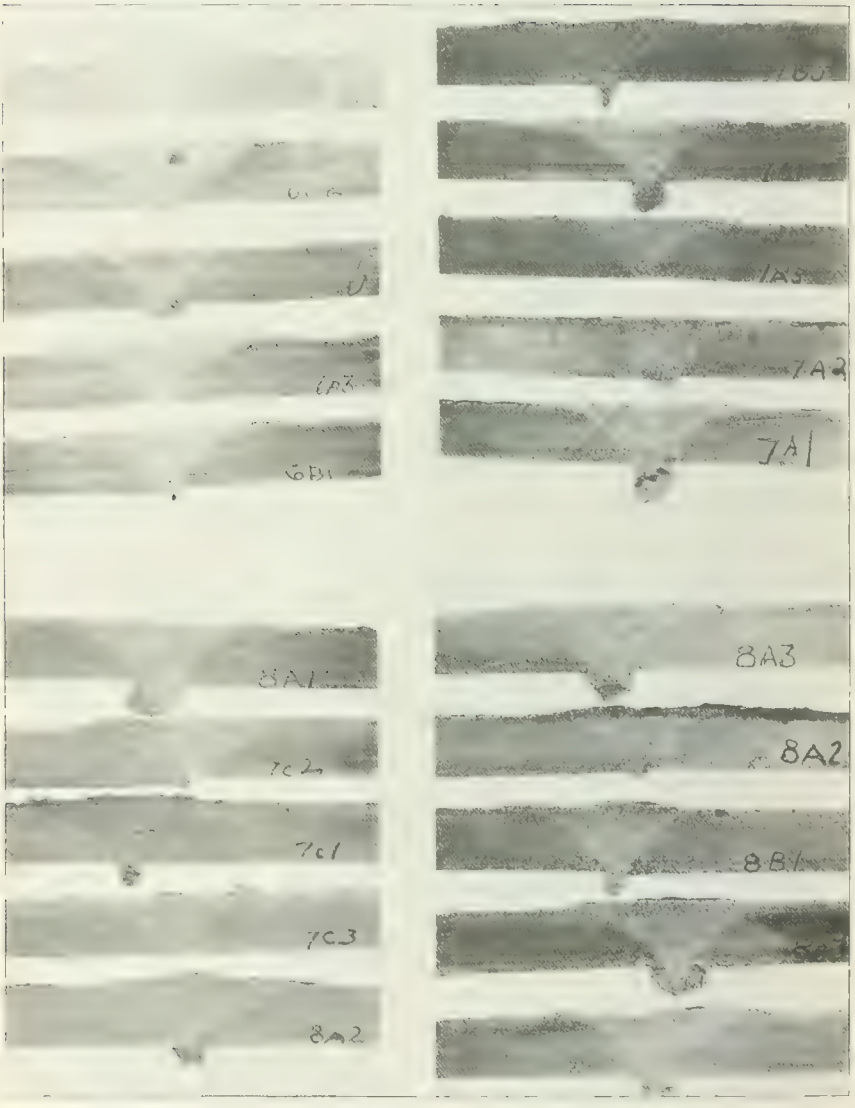


FIG. 68.—*Etched sections through welds*—Continued

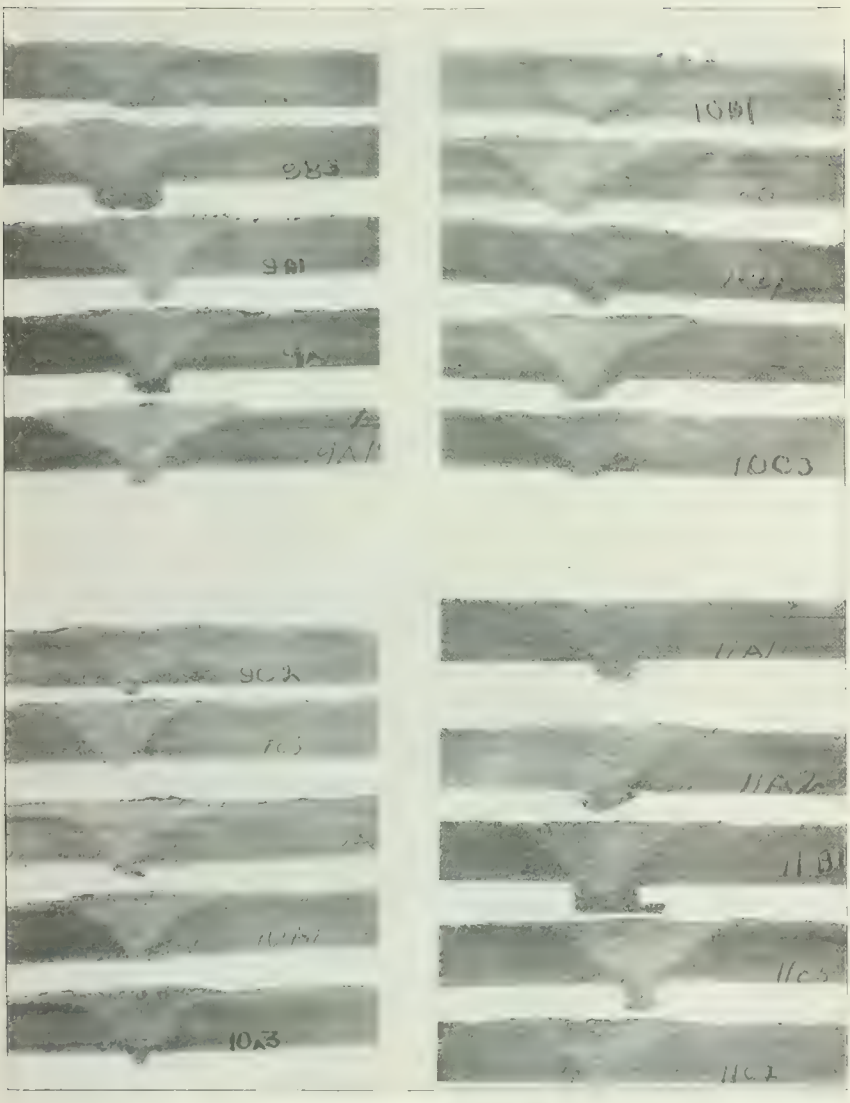


FIG. 69.—Panels after [illegible] (Continued)

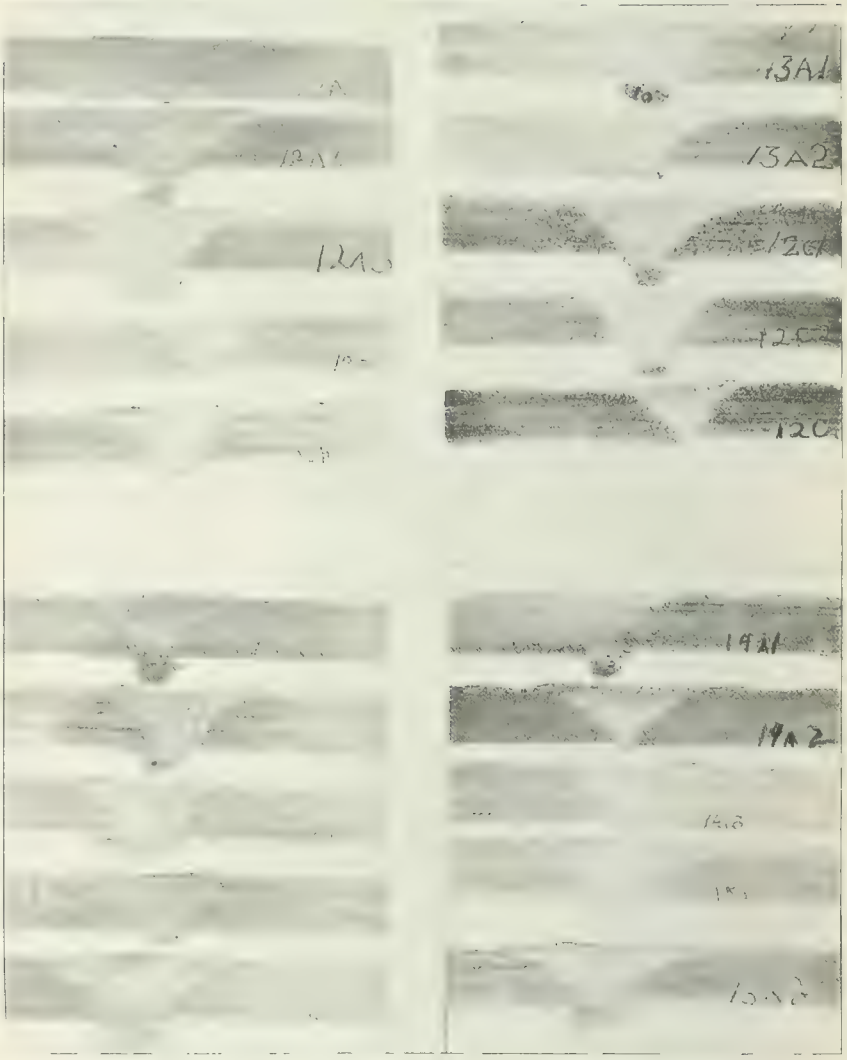


FIG. 50.—*Lateral sections through welds.* Continued

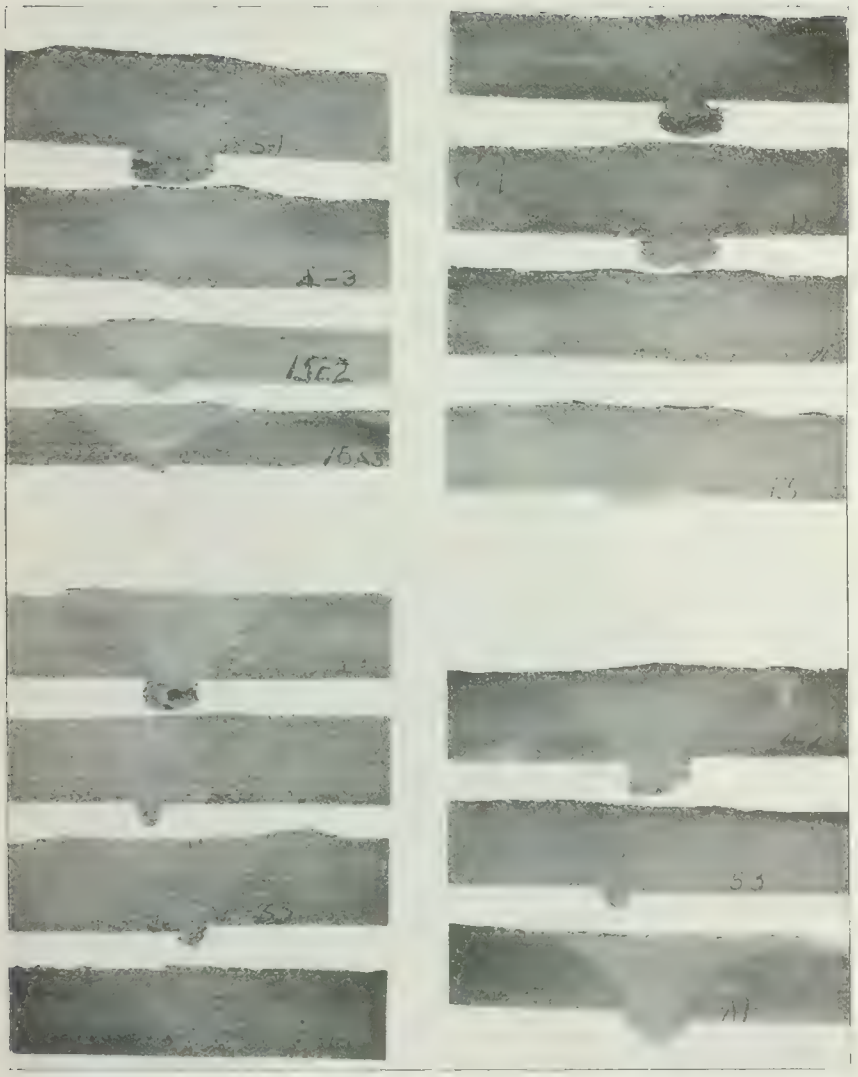


FIG. 71. *Etched sections through welds* — Concluded

possibly this condition may be found to exist for metal of other thicknesses than those used in the tests.

6. That the maximum thickness of metal that may be economically cut with an oxyacetylene blowpipe of standard design when neither the material nor the oxygen is preheated and the cutting is done only from one direction is about 12 inches.

7. That cutting blowpipes, due to their incorrect design, are subject to the same flash-back troubles found in the welding blowpipes.

For the Welding Blowpipes.—1. That the blowpipes most subject to the so-called flash-back phenomena are those in which the oxygen is delivered at a pressure in excess of that at which the acetylene is delivered.

2. That all the blowpipes tested, including those in which the acetylene is delivered at an excess pressure as well as the so-called equal or balanced pressure blowpipes, are subject to flash-back phenomena on account of inherent defects in their design.

3. That the cause of the development of the conditions producing flash back is the setting up within the blowpipe tip and head of a back pressure which retards or chokes off the flow of one of the gases.

4. That this back pressure is the result of confining or restricting the volume flow of the issuing gases at the tip end.

5. That any cause tending to restrict the flow of the gases sets up a back pressure, which immediately causes a change in the amount of each gas delivered to the mixing chamber.

6. That a fluctuating gas volume ratio, due to restriction of volume flow, from whatever cause, prevents a blowpipe from maintaining constantly and at all times during operation the desired "neutral flame."

7. That a blowpipe that can not maintain under all operating conditions a neutral flame can not logically be expected to produce sound welds.

8. That all the blowpipes tested during this investigation, either through improper gas pressures or improper interior design or both are incapable of maintaining a neutral flame (constant volume gas ratio) under all conditions of restricted gas flow and are therefore incapable of producing sound welds where there is any liability of the gaseous products of combustion being momentarily confined, such as occurs in practically all welding operations.

9. That the ability of a blowpipe to consume an equal volume ratio of gases when burning freely and undisturbed in air is no criterion that it is capable of producing sound welds; that is, that it is not subject to detrimental fluctuations in gas ratio during a welding operation and therefore is capable of maintaining a neutral flame under all operating conditions.

10. That whether blowpipes of present designs will consume equal volume ratio of gases when burning freely and undisturbed in air depends on how nearly correct the operator sets the so-called neutral flame, and experience indicates that the average operator checks the acetylene gas flow too much and actually develops an oxidizing rather than a neutral flame.

11. That the question of the possible limiting strength and ductility or the efficiency of welds made by the oxyacetylene welding blowpipe must await the development of a more satisfactory instrument, and that having such an instrument there is no reason to believe that a weld of clean sound metal can not be made with assurance during any welding operation and that such welds will or can be made to possess the proper physical properties.

WASHINGTON, March 13, 1921.

DEPARTMENT OF COMMERCE

TECHNOLOGIC PAPERS

OF THE

BUREAU OF STANDARDS

S. W. STRATTON, DIRECTOR

No. 201

FRICTION AND CARRYING CAPACITY OF BALL AND ROLLER BEARINGS

BY

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Bureau of Standards

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I. INTRODUCTION

In order to facilitate the training of large guns, it is very desirable to reduce the friction at the trunnion bearings. These bearings are moved infrequently and at very low speeds. They may be, however, subjected to great loads when the gun is fired. These conditions are very different from those usual for bearings in engineering work. For the latter the speed is much greater and the periods of operation much longer. They, however, are not often subject to great loads or to impact.

The use of ball and roller bearings for line shafts, vehicle wheels, etc., has become quite extensive, due to their high efficiency. The results obtained from service tests of this kind give very little data for the design of ordnance bearings.

These tests were undertaken by the Bureau of Standards, at the request of the Navy Department, to obtain experimental data on the frictional resistance of both ball and roller bearings at very low speeds and also the loads which they will safely sustain.

The tests may be listed as follows: 1, Static friction test on ball bearing; 2, static friction test on roller bearing; 3, compression test on ball bearing; and 4, compression test on roller bearing.

II. APPARATUS

The special apparatus required for these tests was designed and built by the Navy Department in consultation with the Bureau of Standards. The balls and rollers were obtained from commercial manufacturers and were such as were considered suitable for this use.

1. BALLS

The hardened steel balls were 1.00, 1.25 and 1.50 inches diameter. Four of each size were provided.

2. BALL RACES

The cost of making complete bearings was prohibitive. If, however, complete bearings had been tested, the results could not be used for a bearing having a different diameter, due to the impossibility of measuring the load on the individual balls. Sections of a complete race, only, were represented by small rectangular steel blocks. These are shown in Figs. 1 and 4. Each block had a cylindrical groove on one face, parallel to the opposite face, having a radius slightly greater than that of the ball with which it was to be used. These races were hardened and the groove ground to the required radius. In an actual ball bearing, the axis of the groove would be an arc of a circle about the axis of rotation.

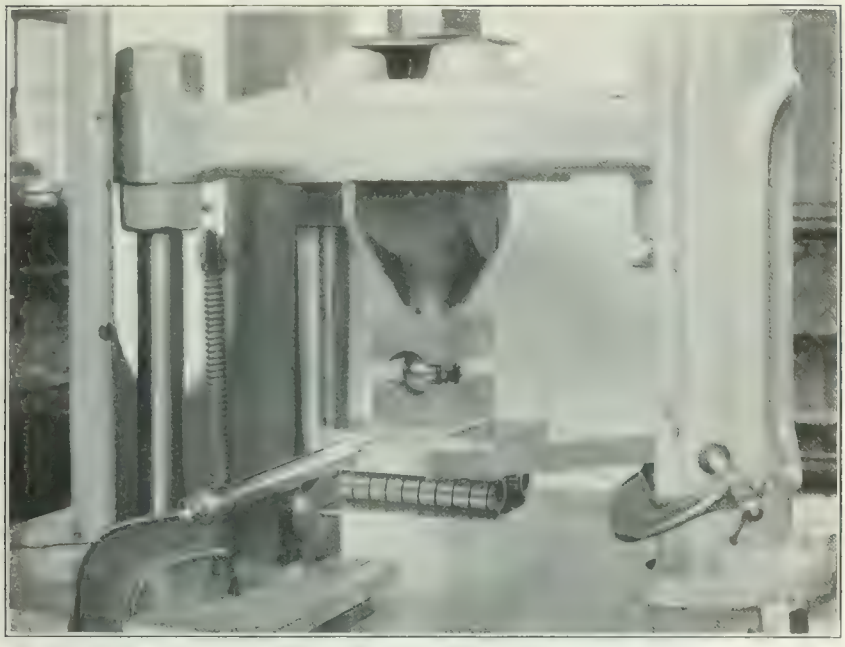

FIG. 1.—*Measuring the static friction of a ball bearing*

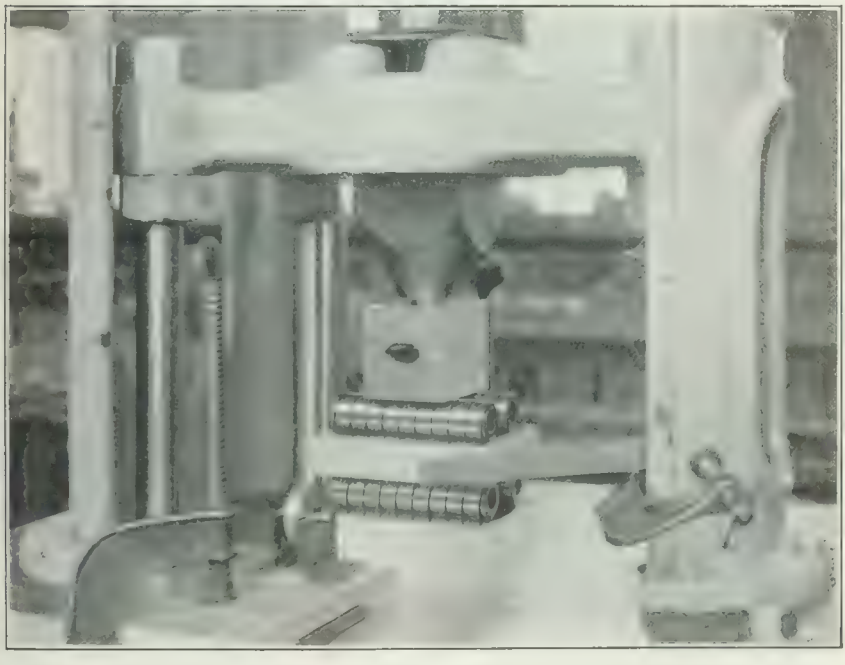

FIG. 2.—*Measuring the static friction of a roller bearing*

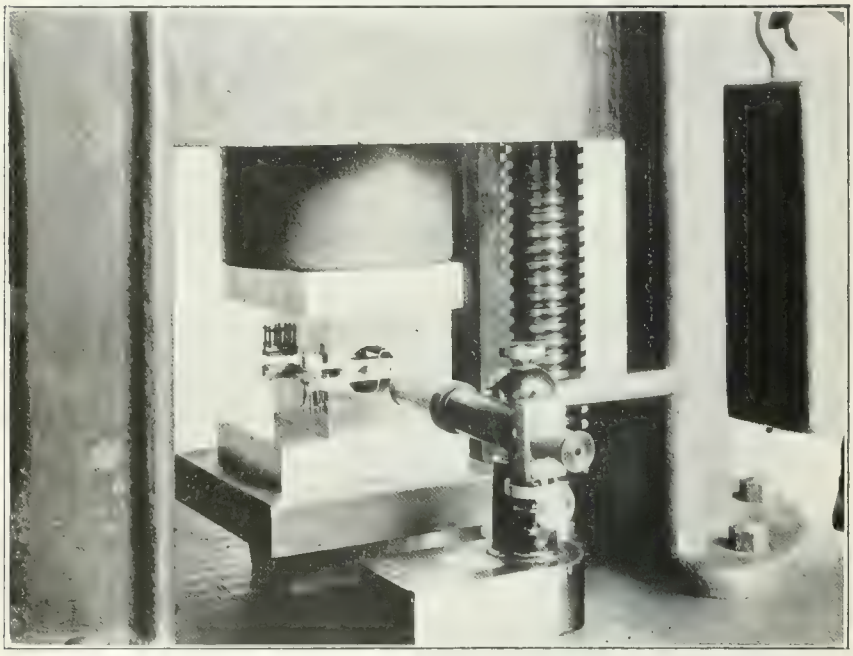

FIG. 3.—*Measuring the deformation of a ball and race under load*

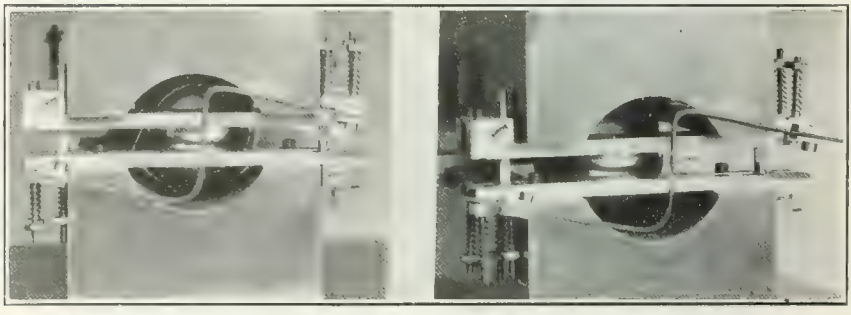

FIG. 4.—*Apparatus for measuring deformation of a ball and race*

The experimental work was much easier because races having straight grooves were used and it is believed that the results apply with reasonable accuracy to bearings having a large diameter such as are used for ordnance work.

Grooved races are used in practice as with them the area of contact between the ball and the race is greater than is obtained with plane races, and therefore the allowable load on the bearing is increased. The load is without doubt a maximum for races grooved to the same diameter as the ball; the friction, however, would be excessive in a bearing of this kind. Two pairs of races were therefore made for each size of ball. One had, perhaps, the smallest practicable radius and the other was somewhat greater. The ratios of groove radii to ball radii are given in Table 1. These races were used both for the friction and the load tests.

TABLE 1.—Ratio of Groove Radii to Ball Radii

Ball diameter, inches	Small groove	Large groove
1.00	1.03	1.10
1.25	1.04	1.12
1.50	1.04	1.12

3. ROLLERS

The rollers were of the flexible roller type. They were closed helices made from steel bars of about 0.52 by 0.30 inch in cross section. The length was about 5.25 inches and the internal diameter about 0.65 inch. They were hardened and the external cylindrical surface ground to about 1.25 inches diameter. These rollers are shown in Fig. 2. Six were provided for these experiments.

4. ROLLER RACES

Two flat plates were used in the roller tests to represent bearings having a large diameter. These are shown in Fig. 2. In order to obtain data also upon bearings such as might be used—for example, for gun trunnions—two segmental bearings having inner diameters of 7 and 20 inches were made. The outer diameter was, of course, greater than the inner diameter by twice the diameter of the rollers. The smaller bearing is shown in Fig. 5. The larger bearing is shown in Figs. 6 and 7. Each of these bearings consisted of the inner race, two portions of the outer race, with apparatus for holding these parts in their proper relative position in a hydraulic testing machine having a capacity of 230 000

pounds. The smaller bearing is shown in the machine in Fig. 8. Side plates furnished bearings for a shaft through the inner race (see Fig. 7) constraining it to rotate about the axis of the bearing. Two rollers, diametrically opposite each other, were used in each of these bearings. As it was found that the rollers tended to become displaced, so that their axes were not parallel to the axis of the bearing, retainers or "cages" were made which rotated about the same shaft as the inner race. One of these cages is shown in Fig. 7. A lever attached to the shaft through the inner race allowed the torque required to rotate the inner race to be measured as shown in Fig. 8.

The bearing surfaces of all flat plates and bearings were hardened and ground.

5. HARDNESS AND DIMENSIONS

The hardness of all bearing parts was measured by the scleroscope, using the universal diamond pointed hammer. The dimensions of the bearing surfaces were also measured. These data are given in Table 2. In the case of the ball races it was found that the ends of grooves were harder than the middle portion of the groove. As the latter portion was used in the experimental work its hardness is given for the average value.

TABLE 2.—Dimensions and Hardness of Balls, Rollers, and Races

Specimen No.	Diameter or radius of curvature	Scleroscope hardness		Specimen No.	Diameter or radius of curvature	Scleroscope hardness	
		Extreme variations of readings	Average			Extreme variations of readings	Average
Diameter of balls	Inches			Diameter of rollers:			
Do	1.0003	62–69	66	1	Inches		
Do	1.2503	57–63	60	2	1.249	71–73	72
Radius of ball races:	1.5000	64–70	68	3	1.249	67–73	70
			At middle of groove	4	1.249	69–71	70
31	.515	62 92	62	5	1.249	67–72	70
32	.515	65–95	65	6	1.250	68–70	69
33	.550	65–86	65	Radius of inner roller races:	1.250	68–73	70
34	.550	60–93	60	43	3.499	80–92	86
35	.650	69–89	69	44	10.000	97–102	99
36	.650	63–90	63	Radius of outer roller races:			
37	.700	70–92	70	39	11.252	70–93	81
38	.700	61–91	61	40	11.252	95–97	96
27	.779	64–67	64	41	4.750	62–70	66
28	.778	60–91	60	42	4.750	71–74	73
29	.839	71 93	71	Plates:			
30	.843	71–91	71	25	Flat within	93–101	97
				26	.0002	94–100	97

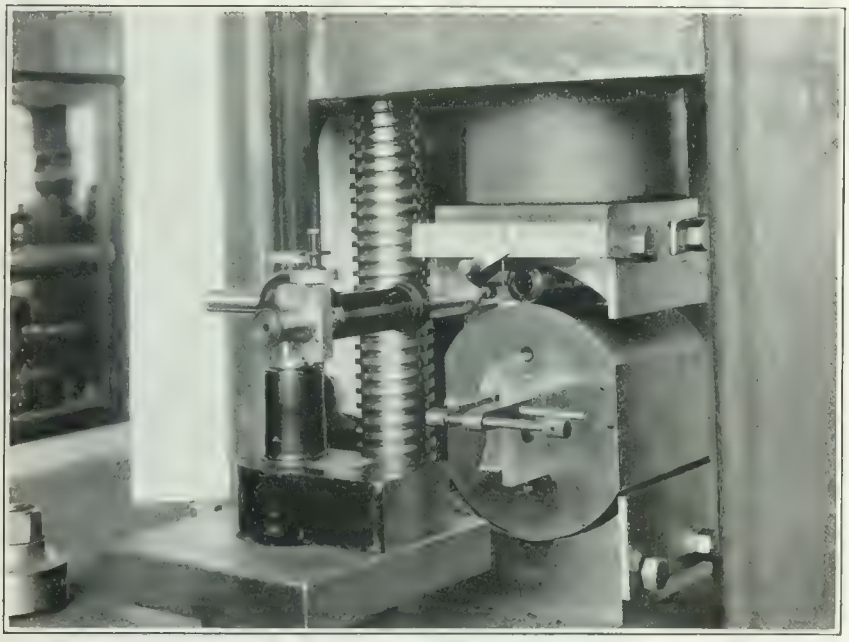

FIG. 5.—*Apparatus for measuring the deformation of a roller in a race having an inner diameter of 7 inches*

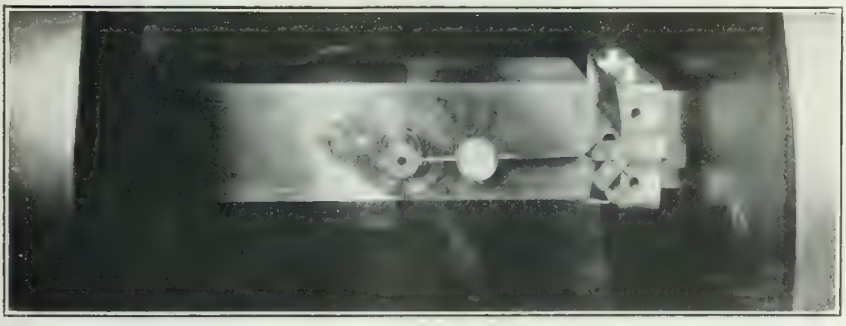

FIG. 6.—*Apparatus for measuring deformation of a roller in a race having an inner diameter of 20 inches*

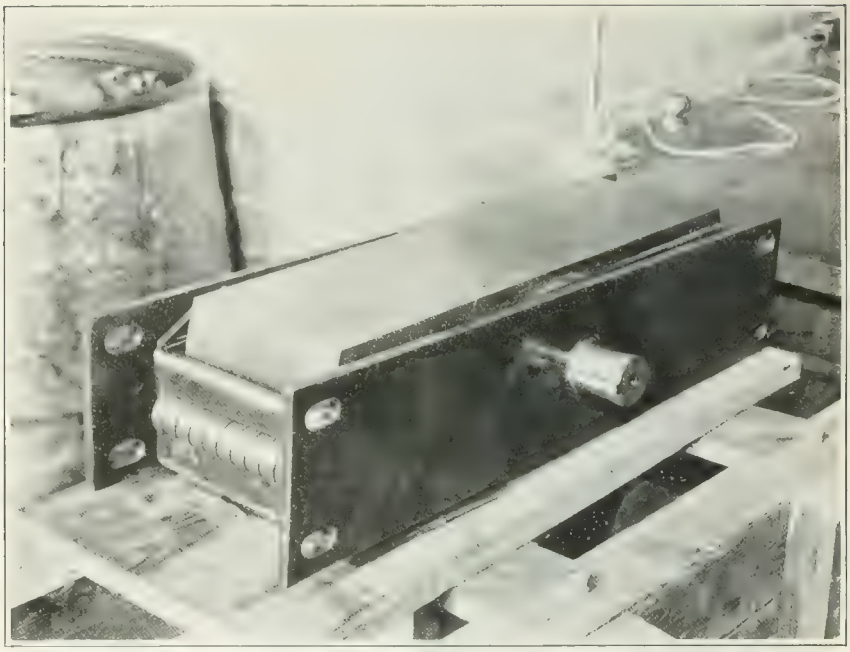

FIG. 7.—*Retainer for roller with inner race*

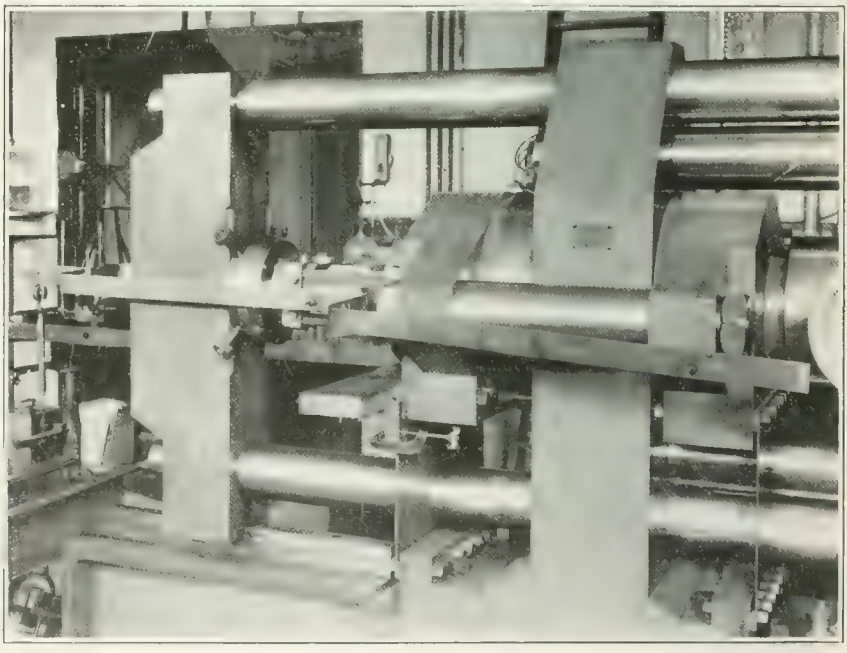

FIG. 8.—*Apparatus for making static friction test of roller and races having an inner diameter of 7 inches*

TABLE 3.—Static Friction of Ball (1 Inch Diameter)

Load on ball, pounds	Radius of races 0.515 inch				Radius of races 0.550 inch			
	Friction, pounds	Ratio friction to load		Coefficient of rolling friction	Friction, pounds	Ratio friction to load		Coefficient of rolling friction
		Observed value	Graph value			Observed value	Graph value	
250	0.11	0.00044	0.00044	0.00022	0.12	0.00048	0.00048	0.00024
500	.23	.00046	.00046	.00023	.31	.00062	.00052	.00026
750	.37	.00049	.00049	.00025	.42	.00056	.00056	.00028
1000	.51	.00051	.00054	.00027	.62	.00062	.00062	.00031
1250	.79	.00063	.00063	.00032	.84	.00067	.00068	.00034
1500	1.19	.00079	.00075	.00038	1.15	.00077	.00078	.00039
1750	1.53	.00087	.00089	.00045	1.63	.00093	.00092	.00046
2000	2.13	.00107	.00105	.00053	2.13	.00107	.00106	.00053
2250	2.81	.00125	.00123	.00062	2.71	.00120	.00120	.00060
2500	3.49	.00140	.00140	.00070	3.40	.00136	.00135	.00068

TABLE 4.—Static Friction of Ball (1.25 Inches Diameter)

Load on ball, pounds	Radius of races 0.650 inch				Radius of races 0.700 inch			
	Friction, pounds	Ratio friction to load		Coefficient of rolling friction	Friction, pounds	Ratio friction to load		Coefficient of rolling friction
		Observed value	Graph value			Observed value	Graph value	
250	0.12	0.00048	0.00044	0.00028	0.14	0.00056	0.00052	0.00032
500	.19	.00038	.00046	.00029	.26	.00052	.00053	.00033
750	.35	.00047	.00049	.00031	.43	.00057	.00054	.00034
1000	.50	.00050	.00051	.00032	.52	.00052	.00055	.00034
1250	.69	.00055	.00053	.00033	.66	.00053	.00057	.00036
1500	.87	.00058	.00057	.00036	.81	.00054	.00061	.00038
1750	1.00	.00057	.00064	.00040	1.16	.00066	.00071	.00044
2000	1.44	.00072	.00076	.00048	1.69	.00084	.00086	.00054
2250	2.10	.00093	.00090	.00056	2.49	.00111	.00106	.00066
2500	2.61	.00104	.00106	.00061	3.21	.00128	.00128	.00080

TABLE 5.—Static Friction of Ball (1.50 Inches Diameter)

Load on ball, pounds	Radius of races 0.779 inch				Radius of races 0.841 inch			
	Friction, pounds	Ratio friction to load		Coefficient of rolling friction	Friction, pounds	Ratio friction to load		Coefficient of rolling friction
		Observed value	Graph value			Observed value	Graph value	
250	0.15	0.00060	0.00055	0.00041	0.13	0.00052	0.00054	0.00041
500	.29	.00058	.00056	.00042	.27	.00054	.00054	.00041
750	.42	.00056	.00056	.00042	.43	.00057	.00055	.00041
1000	.54	.00054	.00056	.00042	.52	.00052	.00055	.00041
1250	.69	.00055	.00057	.00043	.70	.00056	.00056	.00042
1500	.88	.00059	.00057	.00043	.90	.00060	.00056	.00042
1750	1.02	.00058	.00059	.00044	1.02	.00058	.00058	.00044
2000	1.19	.00060	.00064	.00048	1.17	.00059	.00062	.00047
2250	1.65	.00073	.00072	.00054	1.60	.00071	.00069	.00053
2500	2.19	.00088	.00086	.00065	1.95	.00078	.00080	.00060

III. TESTS

1. STATIC FRICTION TEST ON BALL BEARING

(*a*) *Method of Test.*—The arrangements of the apparatus for these tests is shown in Fig. 1. Two balls were used with each pair of races in order to secure stability in the loaded condition. The lower ball race rests upon a plate mounted on two rollers. The upper ball race is loaded by a universal three-screw testing machine having a capacity of 50 000 pounds. A spherical bearing was used between the movable head of the testing machine and the upper ball race. After the desired load had been applied the lower ball race was drawn forward by a force exerted through the spring balance shown which rested on an antifriction roller. The smallest division on the spring balance represented 1 ounce.

The friction of the rollers was found by the method shown in Fig. 2 for each of the loads used for the balls. One-half of the friction for the four rollers was subtracted from the spring balance reading for the ball tests which gave the frictional resistance of the two balls.

In every case the bearings were started from rest. No attempt was made to measure the friction of the bearing after motion occurred, due to the fluctuations in the force and the short distance the bearing could be moved. The starting or static friction is always greater than the moving friction, so that the values given here are in any case the maximum. Care was taken to secure the following conditions during these tests:

1. All bearing surfaces were parallel to each other and also perpendicular to the action line of the load.

2. The balls and rollers were placed symmetrically with relation to the action line of the load.

3. The axes of the rollers were perpendicular to the axis of the ball groove.

4. The action line of the moving force was parallel to the axis of the ball groove.

5. The load was applied equally to the balls and rollers by a spherical bearing block.

It was found that the magnitude of the starting force varied considerably. The load exerted by the testing machine also fluctuated at the instant of starting but rarely more than 50 pounds. These fluctuations may have been due to the following causes:

1. Slight variations in the diameter of the balls and the rollers and variations in the surfaces of the races from the true cylinder or plane.

2. Nonuniform hardness of the bearing surfaces of the races. (The balls and rollers were much more uniform in hardness than the races.)

The conditions under which these tests were made represent ideal rolling friction along a straight line. They are never obtained in practice, so that values in practice may be much larger,

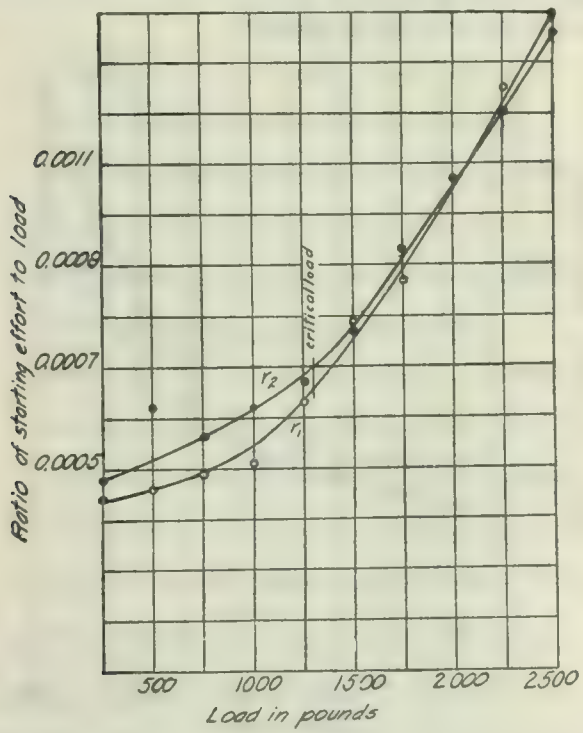

FIG. 9.—*Static friction test on 1-inch ball and races* (r_1=0.515 *inch*, r_2=0.550 *inch*)

due to the sliding friction which occurs. Even in these experiments there was some sliding friction, due to the fact that the area of contact between ball and race, although small, was appreciable. It was also impossible to secure exact arrangement of the parts of the apparatus.

(*b*) *Results.*—The results are given in Tables 3, 4, and 5 and in Figs. 9, 10, and 11. The values given in the tables for the friction are the averages of several trials for slightly different positions of the balls, rollers, and races. The graph values are

obtained from the smooth curve drawn to represent the most probable values.

The coefficients of rolling friction were computed from the graph values by the following formula:[1]

$$\text{Coefficient of rolling friction} = \frac{Pd}{2Q}$$

in which:

$\frac{P}{2}$ = starting friction on one side for one ball or roller, in pounds.

d = diameter of ball or roller in inches.

Q = load on the ball or roller in pounds.

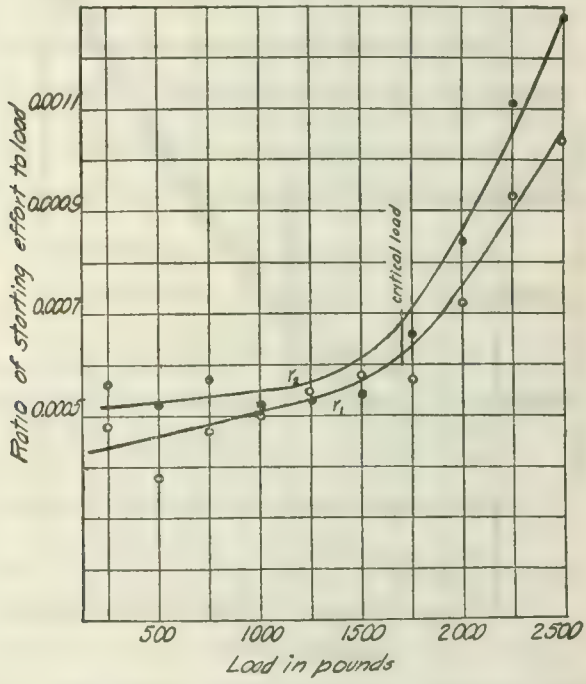

FIG. 10.—*Static friction test on 1¼-inch ball and races (r_1=0.650 inch, r_2=0.700 inch)*

For some of these tests the balls, rollers, and races were well coated with a good mineral lubricating oil. The observed values of the friction, when this was done, appeared to be the same as those obtained when no oil was used.

(*c*) *Conclusions.*—1. The starting friction is nearly the same for both sizes of groove. The groove having the larger radius gave the lowest value for the friction.

[1] R. Thurston, A Treatise on Friction and Lost Work, p. 82, 1885.

2. The ratio of starting friction to the load increases slowly as the load increases, then much more rapidly. The critical loads are approximately as follows:

Ball diameter in inches	Critical load in pounds
1.00	1300
1.25	1700
1.50	2200

If the frictional resistance is to be kept low, these critical loads should not be exceeded. The very rapid rise in the friction at

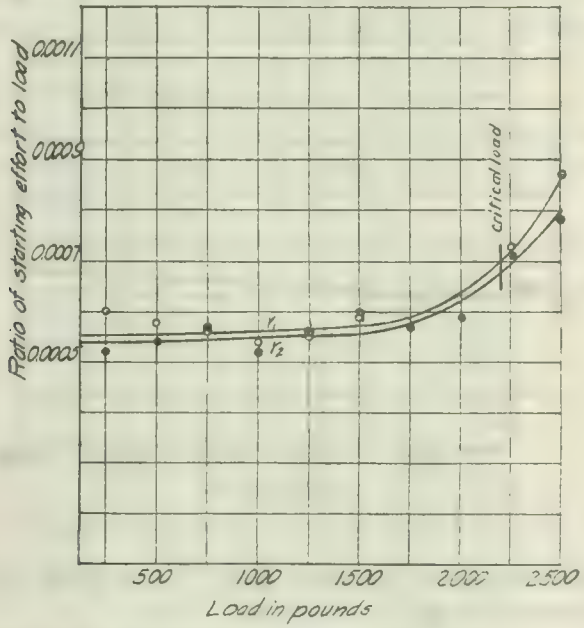

FIG. 11.—*Static friction test on 1½-inch ball and races* (r_1=0.779 *inch*, r_2=0.841 *inch*)

greater loads would seem to indicate that internal work was being performed on the material of either the balls or races which might cause heating and their destruction if the bearings were operated continuously under loads greater than the critical loads.

3. The ratio of frictional resistance to load is practically the same for balls of all diameters up to the critical load and may be taken as 0.00055. For this reason the coefficient of rolling friction as found from the above equation was of little use in these tests.

4. Oil is of little, if any, use upon ball bearings in reducing the static frictional resistance.

2. STATIC FRICTION TEST ON ROLLER BEARING

(*a*) *Method of Test.*—The static friction of the rollers loaded between two *steel plates* was measured as for balls. The arrangement of apparatus is shown in Fig. 2.

The tests of static friction for the two segmental bearings were made in a hydraulic testing machine having a capacity of 230 000 pounds.

The arrangement of the apparatus for the smaller of these bearings is shown in Fig. 8. Two rollers diametrically opposite

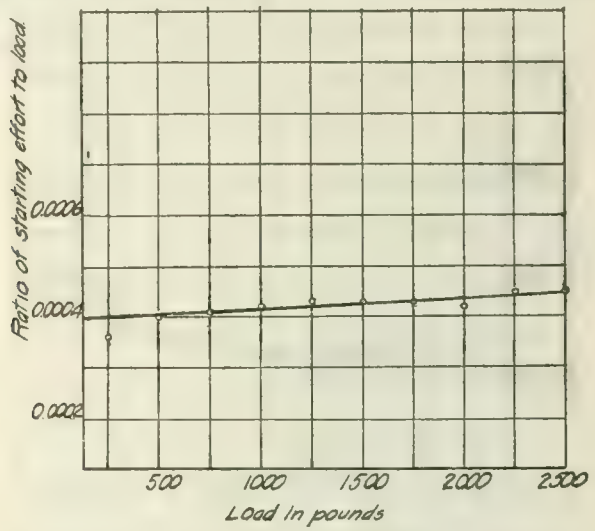

FIG. 12.—*Static friction test of 1¼-inch rollers and plates*

each other were used for each test. These were held in the retainers shown in Fig. 7.

The lever shown in Fig. 8, used for rotating the bearings under load, was 41 inches from the center of rotation to the point of application of the force. This lever was counterbalanced by one of equal length extending in the opposite direction. The force was applied through a spring balance, the smallest graduation of which represented 0.5 pound. Care was taken that the action line of the force was perpendicular to the lever arm. The observed force was used to compute the equivalent frictional force required to cause rotation if applied at the surface of the inner race.

(*b*) *Results.*—The results for these tests are given in Tables 6 and 7 and in Figs. 12 and 30. The values given for the friction are the averages of at least five determinations for each load, as it was found that the friction fluctuated considerably, depending on the position of the rollers with respect to the plane through the

axis of the bearing. This was particularly true with the smaller bearing for which it was very difficult to secure satisfactory readings. This was due probably to the condition of unstable equilibrium of the whole system which existed during these tests and which was beyond the control of the experimenter.

This is the only explanation of the unexpected character of the curve for the smaller bearing in Fig. 30. Several other conditions such as inaccuracies in or nonuniform hardness of the bearing surfaces also affected the friction.

Comparison of the scleroscope hardness values for these bearings as given in Table 2 shows that the smaller bearing averaged about 78, while the larger bearing averaged about 94. It seems very probable that the low hardness values for the small bearing had an important influence on the friction of this bearing.

The coefficient of friction in Table 7 was computed by the formula given above.

TABLE 6.—Static Friction of Roller (1.25 Inches Diameter Between Plates)

Load on roller, pounds	Friction, pounds	Ratio of friction to the load		Coefficient of rolling friction
		Observed value	Graph value	
250	0.09	0.00036	0.000400	0.000250
500	.20	.00040	.000405	.000253
750	.31	.00041	.000410	.000256
1000	.42	.00042	.000415	.000259
1250	.54	.00043	.000420	.000262
1500	.65	.00043	.000425	.000265
1750	.75	.00043	.000430	.000268
2000	.85	.00042	.000435	.000271
2250	1.01	.00045	.000440	.000274
2500	1.14	.00046	.000445	.000277

TABLE 7.—Static Friction of Roller (1.25 Inches Diameter)

Load on roller, pounds	Radius of inner race 3.5 inches			Radius of inner race 10.0 inches		
	Friction, pounds	Coefficient of rolling friction		Friction, pounds	Coefficient of rolling friction	
		Observed value	Graph value		Observed value	Graph value
5 000	8.8	0.00110	0.00110	6.1	0.00076	0.00075
10 000	29.3	.00183	.00190	12.3	.00077	.00077
15 000	58.5	.00244	.00245	19.5	.00081	.00080
20 000	87.8	.00275	.00275	26.6	.00083	.00083
25 000	120.0	.00300	.00305	34.8	.00087	.00091
30 000	170.0	.00354	.00330	53.3	.00110	.00105
40 000	234.4	.00366	.00378	92.3	.00144	.00145
50 000	316.5	.00396	.00420	153.7	.00192	.00192

(*c*) *Conclusions.*—Consideration of the values for the coefficient of rolling friction for the bearing having a radius of 10 inches shows that the static friction is nearly constant up to a load of 25 000 pounds. For greater loads the friction increases rapidly. This is similar to the behavior of the balls, and it is believed that this critical load should be considered the allowable load on the roller.

Due to the unexpected character of the curve the critical load for the bearing having an inner diameter of 7 inches could not be determined.

The critical loads as obtained from the load friction diagram (Fig. 30) are approximately as follows:

Radius of inner races in inches	Critical load in pounds
10.0	25 000
3.5	

3. COMPRESSION TEST ON BALL BEARING

(*a*) *Method of Test.*—The allowable load on a bearing may be determined by noting the greatest load which it will sustain without permanent deformation. (See Tables 8, 9, and 10.) The apparatus for this test was that used for the friction tests but arranged as shown in Fig. 3. A single ball was placed between the races and the load applied by the testing machine previously used.

(*b*) *Compression and Set.*—As it was impossible to measure the deformation of the ball under load, special apparatus was designed to measure the relative motion of the two races; that is, the deformation of balls and races combined. This apparatus is shown in Figs. 3 and 4. At each corner of the races is a steel rod secured to one race. Opposite it is a short steel lever carried by a horizontal shaft which is held in any position in which it may be placed by caps for the bearing loaded by long helical springs. Experience with this apparatus showed that the best results were obtained when the shaft rested in a triangular groove in the supports. The caps for the bearings were also grooved but were later turned over to present a plane surface to the shaft which was, therefore, held in a three-line bearing.

The end of the shaft which projects from the bearing carries a curved pointer, the end of which opposes the end of the pointer on the other side of the races. In Fig. 4, the rod secured to the upper race is seen at the left and the one secured to the lower race at the right. The levers are not visible but the pointers are clearly shown.

In use, the pointers are turned away from each other, the desired load is applied to the bearing, then the pointers are turned toward each other by hand so that each lever comes in contact with the corresponding rod. The distance between the two pointers is then measured by the micrometer microscope shown in Fig. 3. The arrangement of this apparatus is such as to give correct values, even if the races are slightly tilted during the test. The total deformation of ball and race combined under load may be obtained as well as the permanent deformation after removing the load. The pointers multiplied the movement of the levers 10 times. The arrangement of the pointers, in pairs, made the change in distance between pointers 20 times the change in the distance between the races.

TABLE 8.—Compression Test of Ball (1 Inch Diameter)

Load in pounds	Radius of races	Total deformation of ball and races			Permanent set of ball and races		Contact area			
		Observed value	Graph value	Hertz value	Observed value	Graph value	2a	2b	2b (Hertz value)	Area
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch²
500	0.515	0.00079	0.00088	0.00112	0.00003	0.00002				
	.550	.00097	.00099	.00112	.00003	.00003				
	.779									
1000	.515	.00140	.00156	.00179	.00006	.00006	0.292	0.038	0.040	0.0087
	.550	.00170	.00172	.00177	.00007	.00007	.188	.043	.040	.0058
	.779						.098	.051		.0039
	∞						.056	.056	.056	.0025
1500	.515	.00212	.00213	.00234	.00013	.00012				
	.550	.00231	.00233	.00232	.00014	.00014				
	.779									
2000	.515	.00265	.00264	.00283	.00021	00021	.360	.050	.050	.0141
	.550	.00283	.00287	.00282	.00023	.00024	.245	.058	.052	.0112
	.779						.122	.070		.0067
	∞						.073	.073	.072	.0042
2500	.515	.00314	.00310	.00329	.00033	.00032				
	.550	.00336	.00336	.00327	.00038	.00037				
	.779									
3000	.515	.00355	.00352	.00371	.00048	.00047	.397	.057	.058	.0178
	.550	.00382	.00382	.00369	.00054	.00053	.273	.067	.058	.0143
	.779						.134	.080		.0084
	∞						.083	.083	.082	.0054
3500	.515	.00397	.00393	.00412	.00067	.00067				
	.550	.00425	.00425	.00409	.00075	.00077				
	.779									
4000	.515						.423	.062	.064	.0206
	.550						.288	.073	.064	.0165
	.779						.141	.084		.0093
	∞						.089	.089	.090	.0062

Two microscopes, one at each end of the race, were used by which a difference in the distance between the pointers of 0.00004 inch could be observed by estimation. The displacement of either end of the ball race could therefore be measured within 0.000004 inch.

TABLE 9.—Compression Test of Ball (1.25 Inches Diameter)

Load in pounds	Radius of races	Total deformation of ball and races			Permanent set of ball and races		Contact area			
		Observed value	Graph value	Hertz value	Observed value	Graph value	2a	2b	2b (Hertz value)	Area
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch²
500	0.650	0.00081	0.00085	0.00104	0.00002	0.00002				
	.700	.00063	.00087	.00104	.00002	.00003				
	.779									
1000	.650	.00154	.00150	.00166	.00005	.00005	0.283	0.044	0.044	0.0098
	.700	.00142	.00157	.00165	.00005	.00007	.183	.048	.044	.0069
	.779						.144	.053		.0060
	∞						.058	.058	.062	.0027
1500	.650	.00207	.00207	.00217	.00009	.00008				
	.700	.00207	.00217	.00217	.00009	.00011				
	.779									
2000	.650	.00254	.00253	.00263	.00014	.00014	.367	.056	.054	.0161
	.700	.00265	.00268	.00261	.00015	.00017	.237	.063	.056	.0117
	.779						.182	.070		.0100
	∞						.075	.075	.076	.0044
2500	.650	.00299	.00299	.00303	.00021	.00021				
	.700	.00314	.00317	.00303	.00023	.00026				
	.779									
3000	.650	.00339	.00340	.00342	.00030	.00030	.409	.063	.062	.0202
	.700	.00362	.00363	.00342	.00035	.00037	.265	.072	.064	.0150
	.779						.205	.079		.0126
	∞						.086	.086	.088	.0058
3500	.650	.00381	.00378	.00382	.00042	.00043				
	.700	.00409	.00407	.00379	.00050	.00050				
	.779									
4000	.650						.432	.069	.070	.0234
	.700						.282	.078	.070	.0172
	.779						.220	.085		.0146
	∞						.094	.094	.096	.0069

(*c*) *Contact Area.*—Several different methods were tried of making visible the area of contact between the ball and the race. The one which was best suited for the purpose and was, therefore, used in this work was a thin film of lubricating oil on the surface of the race. This film applied with the fingers, which were used to wipe the surface almost dry, was extremely thin. The ball was well cleaned.

TABLE 10.—Compression Test of Ball (1.5 Inches Diameter)

Load in pounds	Radius of races	Total deformation of ball and races			Permanent set of ball and races		Contact area			
		Observed value	Graph value	Hertz value	Observed value	Graph value	2a	2b	2b (Hertz value)	Area
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch ^{2}
500	0.779	0.00071	0.00072	0.00098	0.00002	0.00002				
	.841	.00078	.00082	.00098	.00003	.00003				
1000	.779	.00126	.00131	.00156	.00005	.00005	0.270	0.045	0.046	0.0096
	.841	.00149	.00147	.00155	.00005	.00006	.189	.049		.0075
	∞						.059	.059	.064	.0027
1500	.779	.00177	.00182	.00204	.00007	.00007				
	.841	.00203	.00205	.00203	.00009	.00009				
2000	.779	.00225	.00228	.00247	.00011	.00012	.373	.058	.058	.0170
	.841	.00256	.00257	.00246	.00014	.00014	.249	.065		.0127
	∞						.080	.080	.082	.0050
2500	.779	.00272	.00271	.00287	.00016	.00016				
	.841	.00305	00303	.00285	.00021	.00020				
3000	.779	.00311	.00311	.00324	.00022	.00022	.427	.065	.066	.0218
	.841	.00348	.00349	.00322	.00028	.00028	.285	.075		.0168
	∞						.094	.094	.094	.0069
3500	.779	.00349	.00348	.00359	.00030	.00029				
	.841	.00395	.00393	.00357	.00038	.00038				
4000	.779						.458	.072	.074	.0258
	.841						.309	.083		.0201
	∞						.103	.103	.104	.0083

TABLE 11.—Compression Test of Balls

Load on ball, in pounds	Total deformation of ball and races					
	Ball, 1 inch diameter; radius of races, 0.550 inch		Ball, 1.25 inch diameter; radius of races, 0.700 inch		Ball, 1.5 inch diameter; radius of races, 0.841 inch	
	Observed value	Graph value	Observed value	Graph value	Observed value	Graph value
	Inch	Inch	Inch	Inch	Inch	Inch
500		0.00080		0.00080		0.00080
4 000	0.00376	.00445	0.00389	.00430	0.00348	.00420
8 000	.00671	.00753	.00679	.00740	.00618	.00690
12 000	.00950	.01040	.00932	.01010	.00844	.00930
16 000	.01231	.01320	.01167	.01250	.01066	.01150
20 000	.01528	.01600	.01367	.01475	.01277	.01360
24 000	.01790	.01870	.01598	.01695	.01482	.01570
28 000	.02087	.02145	.01834	.01915	.01698	.01775
32 000			.02070	.02125	.01806	.01875

The area of contact between the race and the ball was distinctly visible, as it appeared darker than the surrounding surface.

The edges of this area were sharply defined. The thickness of the oil film was estimated by drawing a ball lightly across an oiled plate and measuring the width of the dark band. Knowing the diameter of the ball, the angle subtended by the band at the center of the ball was easily computed, and from this the versine of half this angle. This, multiplied by the radius of the ball, was assumed to be the thickness of the oil film. The area of contact was measured by means of a microscope reading (by estimation) to 0.0004 inch.

After applying the load to the ball resting on the oiled surface, the ball was removed and the total area of contact was computed.

(*d*) *Results.*—The results of these tests are given in Tables 8, 9, 10, and 11. In the tables are also given the values of the deformations and of the areas of contact calculated by Hertz's theory.[2] Hertz's results may be written:

$$a = \mu \sqrt[3]{\frac{P}{H}\delta}$$

$$b = \nu \sqrt[3]{\frac{P}{H}\delta}$$

$$\alpha = \xi \sqrt[3]{\left(\frac{P}{H}\right)^2 \frac{1}{\delta}}$$

$$2(A+B) = \frac{4}{\delta} = \zeta_{11} + \zeta_{12} + \zeta_{21} + \zeta_{22}$$

$$2(A-B) = +\sqrt{(\zeta_{11}-\zeta_{12})^2 + (\zeta_{21}-\zeta_{22})^2 + 2(\zeta_{11}-\zeta_{12})(\zeta_{21}-\zeta_{22})\cos 2\omega}$$

$$\cos \tau = \frac{A-B}{A+B}, \quad H = \frac{4}{3}\,\frac{E}{1-\delta^2}$$

where:

α = total deformation of ball and races combined

$\left.\begin{matrix} 2a \\ 2b \end{matrix}\right\}$ = diameters of area of contact

P = load

E = Young's modulus = 30 000 000 lbs./in.^{2}

δ = Poisson's ratio = 3/10

H = 44 000 000 lbs./in.^{2}

$\zeta_{11}, \zeta_{12}; \zeta_{21}, \zeta_{22}$ are the reciprocals of the principal radii of curvature of the two bodies; ω, the angle between their principal planes

[2] Heinrich Hertz, Gesammelte Werke, Leipzig 1895, 1, pp. 155 to 173; and F. Heerwagen, Zeitschrift des Vereins deutscher Ingenieure, 45, pp. 1701 to 1705; 1901.

of curvature and μ, ν and ξ, transcendental functions of the auxiliary angle τ, expressed in terms of elliptic integrals. μ, ν and ξ have been taken from the tables of Hertz and Heerwagen and are given below in Table 12 which was prepared by Dr. L. B. Tuckerman.

TABLE 12.—Coefficients for Hertz's Theory

τ	μ	ν	ξ	τ	μ	ν	ξ
30 degrees	2.731	0.493	1.453	70 degrees	1.284	0.802	1.944
35 degrees	2.397	.530	1.550	75 degrees	1.202	.846	1.967
40 degrees	2.136	.567	1.637	80 degrees	1.128	.893	1.985
45 degrees	1.926	.604	1.709	85 degrees	1.061	.944	1.996
50 degrees	1.754	.641	1.772	90 degrees	1.000	1.000	2.000
55 degrees	1.611	.678	1.828	95 degrees	.944	1.061	1.996
60 degrees	1.486	.717	1.875	100 degrees	.893	1.128	1.985
65 degrees	1.378	.759	1.912				

The values of total deformation approach closely those given by theory as shown in Figs. 20, 21, and 22. The existing differences may be explained by the nonuniform hardness, the difference between the actual and the assumed elastic properties of the material, and in addition by the fact that the major diameter of the area of contact is not as assumed by the theory, very small in comparison with the diameter of the ball. The same is true for the area of contact.

These tests show that the radii of the races influence the amount of the total deformation and of the permanent set more than the theory would indicate and in the opposite direction, that is, the larger the radii of races, the greater the deformation.

The total deformation of the ball was not measured separately but the direct measurements of the set of the races and the ball showed that the permanent set of a ball even for a load of 30 000 pounds does not exceed 0.00020 inch for 1½-inch diameter ball nor 0.00015 inch for a 1-inch diameter ball. Thus the permanent set observed is due almost exclusively to the races. The carrying capacity of balls with races given in Tables 13 and 14 are therefore limited by the deformation of the races. If the races had been harder, the values would have been higher. The theoretical value of 2a (the major diameter of contact area) is not given in the tables since it is so large that even approximate agreement could not be expected.

The values for the area of contact are plotted in Figs. 13, 14, 15, 16, and 17. Those for the deformation are shown in Figs. 20, 21, 22, and 23. The tests showed that even up to very high loads, far beyond those actually used in practice, the law of strains does not undergo any sharp change. The total deformation of ball and races follows pretty closely the law of a straight line with only a slight tendency to decrease gradually with an increase of load. The permanent set follows, also, the law of a

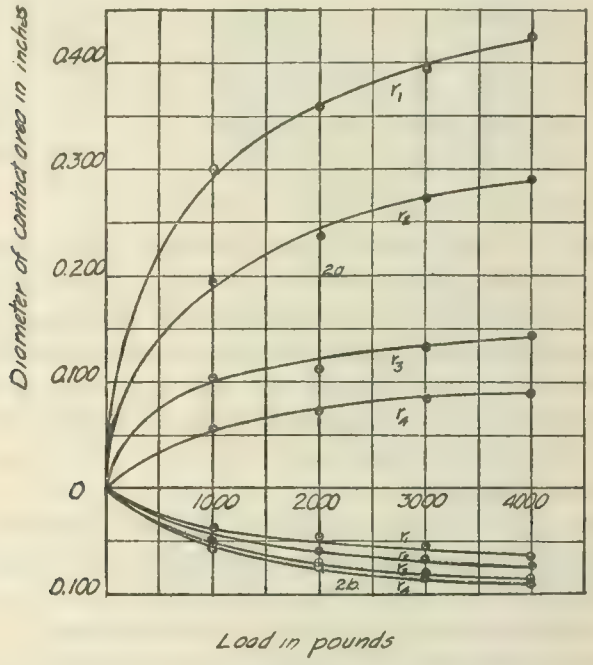

FIG. 13.—*Area of contact of 1-inch ball and races* (r_1=0.515 *inch*, r_2=0.550 *inch*, r_3=0.770 *inch*, $r_4=\infty$)

straight line but tends to increase gradually with an increase of load.

(*e*) *Conclusions.*—The allowable load on balls, as far as the permanent set is concerned, is limited to the load, which if increased, will produce a permanent set of either the balls or races, which would cause the bearing to fail to function properly. The permanent set will, in practice, first occur, probably, in the races. As the permanent set of the races grows very gradually, there is no definite indication of this load limit so that any limit selected is more or less arbitrary.

If we select 0.0001 inch [3] as the allowable permanent set of a race, we have from these tests the values of Table 13 for the carrying capacities of balls.

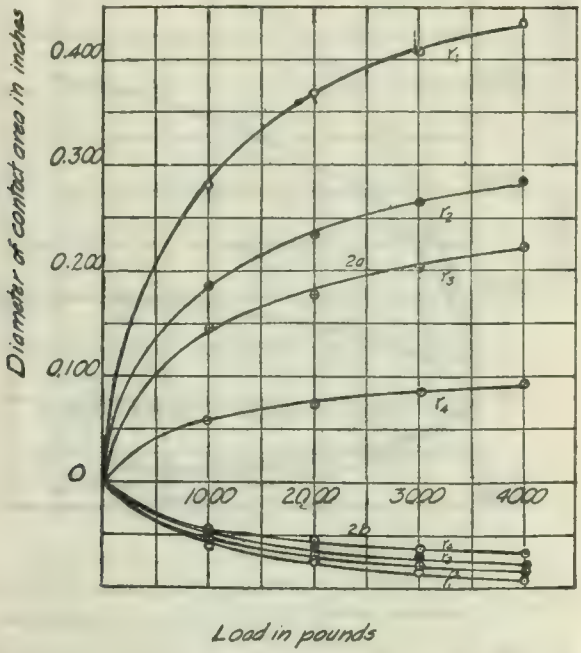

FIG. 14.—*Area of contact of 1¼-inch ball and races (r_1=0.650 inch, r_2=0.700 inch, r_3=0.779 inch, $r_4=\infty$)*

TABLE 13.—Carrying Capacities of Balls with Races

Diameter of ball	Radius of race		Allowable load	
	r_1	r_2	r_1	r_2
	Inch	Inch	Pounds	Pounds
1.00 inch	0.515	0.550	2000	1800
1.25 inches	.650	.700	2500	2300
1.50 inches	.779	.841	2800	2500

A comparison of these values, with those given by the static friction test, shows that they are about 30 per cent larger. The allowable load on a ball may also be computed from the formula, $P=cd^2$, derived by Prof. Stribeck,[4] in which P is the load on the ball in kilograms; d is the diameter of the ball in centimeters,

[3] This value is often used as the allowable variation in the diameter of balls for bearings.

[4] Zeitschrift des Vereines deutscher Ingenieure, 45, p. 791, 1901.

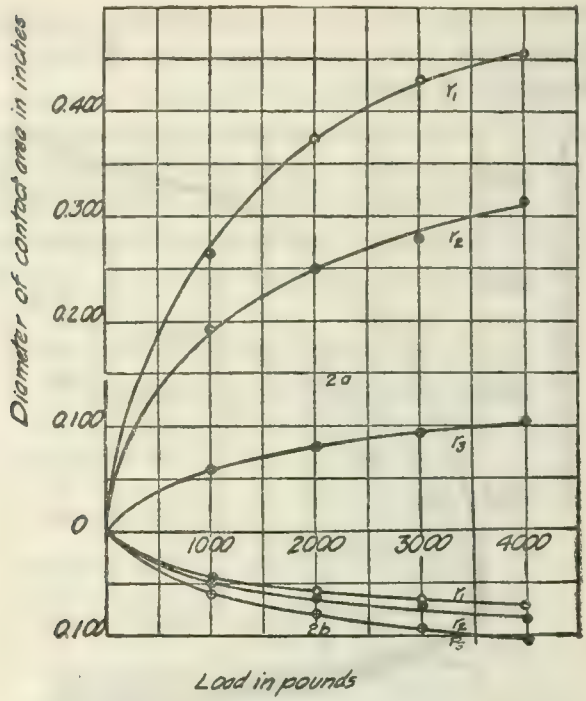

FIG. 15.—*Area of contact of 1½-inch ball with races* (r_1=0.779 *inch*, r_2=0.841 *inch*, r_3=∞)

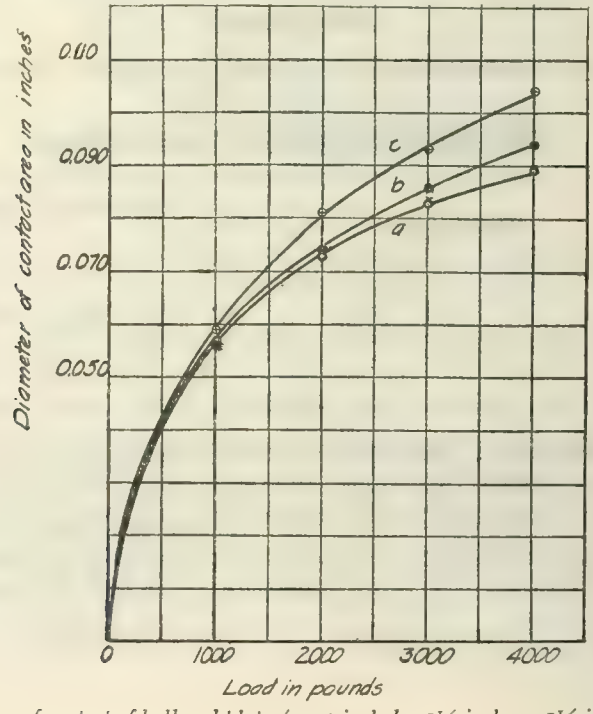

FIG. 16.—*Area of contact of ball and plates* (*a*=1-*inch*, *b*=1¼-*inch*, *c*=1½-*inch diameter*)

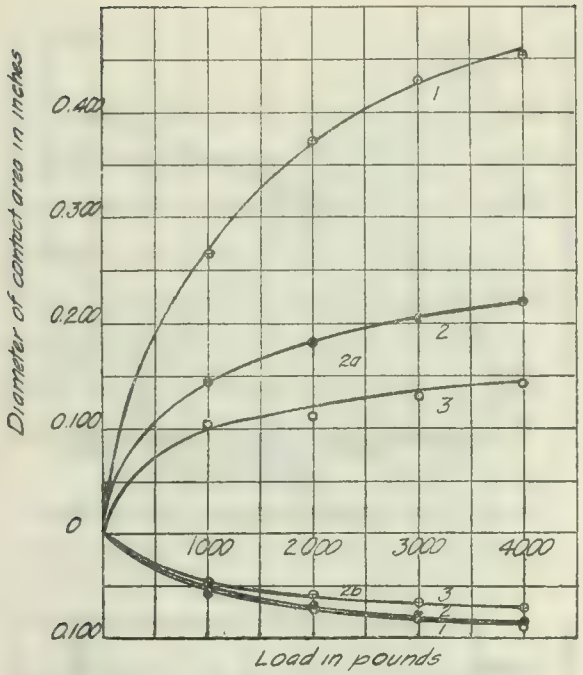

FIG. 17.—*Area of contact of ball and races of radius 0.779 inch*

Curve 1 for 1½-inch ball, curve 2 for 1¼-inch ball, and curve 3 for 1-inch ball

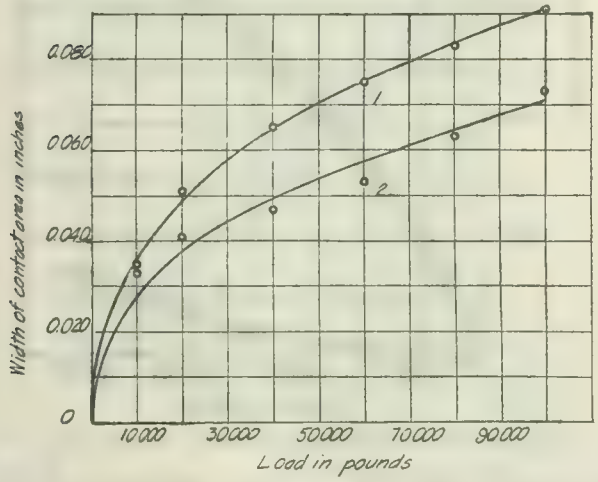

FIG. 18.—*Area of contact of a 1¼-inch roller between races of 3.5 inches and of 4.75 inches*

Curve 1, outer race; curve 2, inner race

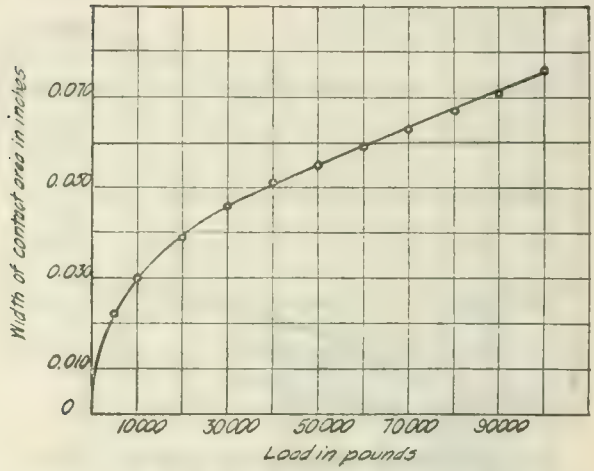

FIG. 19.—*Area of contact of 1¼-inch roller and plates*

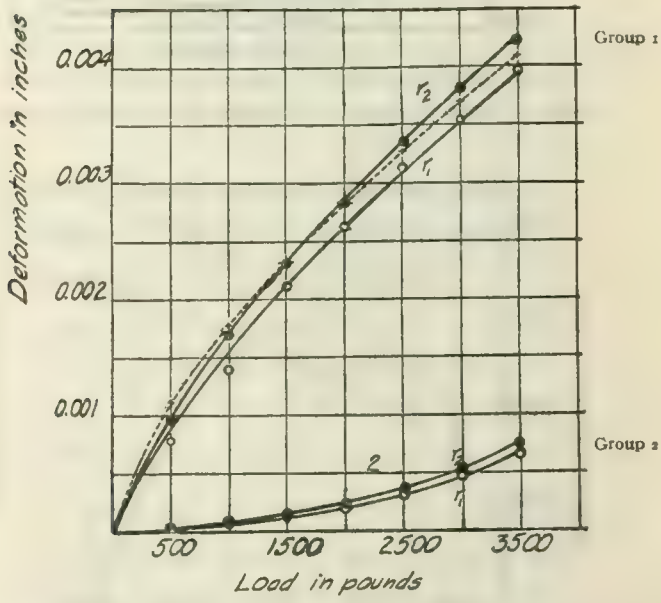

FIG. 20.—*Compression test on 1-inch ball with races* (r_1=0.515 *inch*, r_2=0.550 *inch*)

Curves in group 1 show total deformation; curves in group 2 show permanent set. The dotted line shows Hertz's values

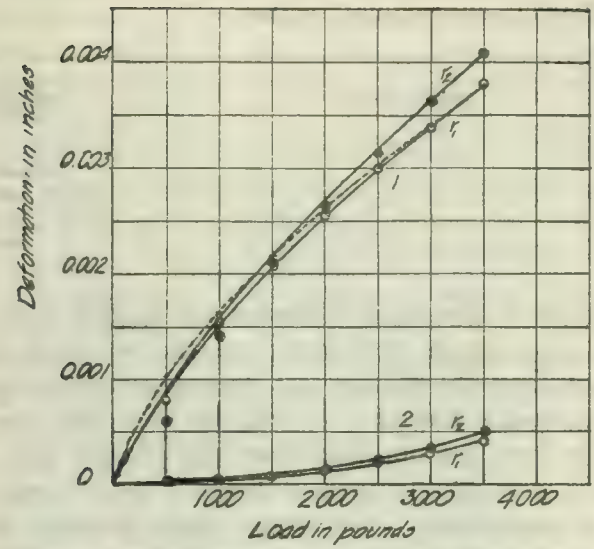

FIG. 21.—*Compression test on 1¼-inch ball with races (r_1=0.650 inch, r_2=0.700 inch)*

Curves in group 1 show total deformation; curves in group 2 show permanent set. The dotted line shows Hertz's values

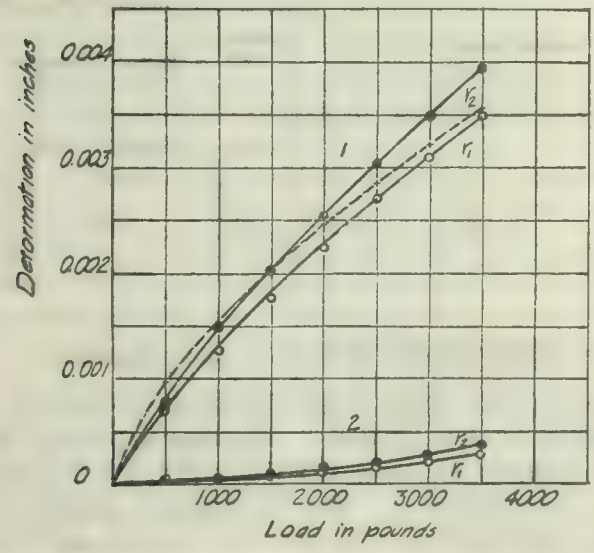

FIG. 22.—*Compression test on 1½-inch ball and races (r_1=0.779 inch, r_2=0.841 inch)*

Curves in group 1 show total deformation; curves in group 2 show permanent set. The dotted line shows Hertz's values

and c is a constant depending on the material. This formula gives the following approximate values:

Diameter of ball	C=100 Allowable load P	C=150 Allowable load P
	Pounds	Pounds
1.00 inch	1400	2100
1.25 inches	2200	3300
1.50 inches	3200	4800

The values for P have been converted into English units.

In Table 14 are given, for comparison, the values of allowable load, as found from the friction test, compression test, and those found by Stribeck's formula. It will be seen that the lowest values of the load are obtained from the friction test. These values should, probably, be used in design if the efficiency of the bearing is of importance. The larger values obtained from the compression tests may be, however, used before rapid deterioration of the bearings will result.

TABLE 14.—Carrying Capacities of Ball Bearings

Diameter of ball	Radius of races	Allowable load, ball with races		
		Friction test	Compression test	Stribeck formula (c=100)
	Inch	Pounds	Pounds	Pounds
1 inch	0.515	1300	2000	1400
	.550		1800	
1.25 inches	.650	1700	2500	2200
	.700		2300	
1.5 inches	.779	2200	2800	3200
	.841		2500	

4. COMPRESSION TEST ON ROLLER BEARING

(*a*) *Method of Test.*—These tests were made in the same manner as the compression tests for balls. The arrangement of the apparatus for the compression tests with the bearing having the smaller diameter is shown in Fig. 5. The two opposed pointers attached to the outer and inner race, respectively, were used to measure the deformation of the roller and races combined. A micrometer microscope was used at both ends of the roller to measure the distance between the ends of the pointers. The load was applied with a testing machine having a capacity of 100 000 pounds.

The compression tests with the bearings having the larger radius were made in a hydraulic testing machine having a capacity of 2 300 000 pounds in compression. The apparatus is shown in Fig. 6. Two dial micrometers were used to measure the deformation. The smallest division of these micrometers is 0.001 inch and fifths of a division could be estimated. A similar arrangement was used in testing the rollers between plates and the same testing machine and measuring apparatus were used. With this apparatus some compression tests were carried beyond the elastic limit of the rollers and, from the stress diagrams, the proportional limit was obtained.

(*b*) *Results.*—The data for the compression tests of rollers are given in Tables 15, 16, 17, and 18. The deformations are in each case the values for both roller and race. The theoretical values given in the tables are computed according to the formula of Hertz given above. The results are plotted in Figs. 24, 25, and 26, which show the relation of the deformation to the load. Figs. 18, 19, and 27 show the relation of area of contact to the load. The stress diagrams are shown in Figs. 28 and 29.

Inspection of the rollers and races showed that unlike the results with ball bearings the permanent set of the races was quite negligible compared with the permanent set of the rollers. Measurements of the diameters of a roller which had been broken under compressive loading show that the diameter at the middle of the length of the roller parallel to the line of application of the force was reduced, that perpendicular to the action line of the force it was increased. This was to be expected. Both these diameters at the ends of the rollers were reduced. This behavior seems to show that the ends of the rollers twist under load so as to decrease the diameter. It follows that the ends of a "flexible" roller carry less load than the middle portion.

(*c*) *Conclusions.*—The maximum load for a flexible roller (1.25 inches diameter and 5.25 inches long) is 135 000 pounds. This is the proportional limit for these rollers. It is believed that this value tends to become smaller as the radius of the races increases. It should be noted that the critical load found from the friction tests was only 25 000 pounds, a much lower value.

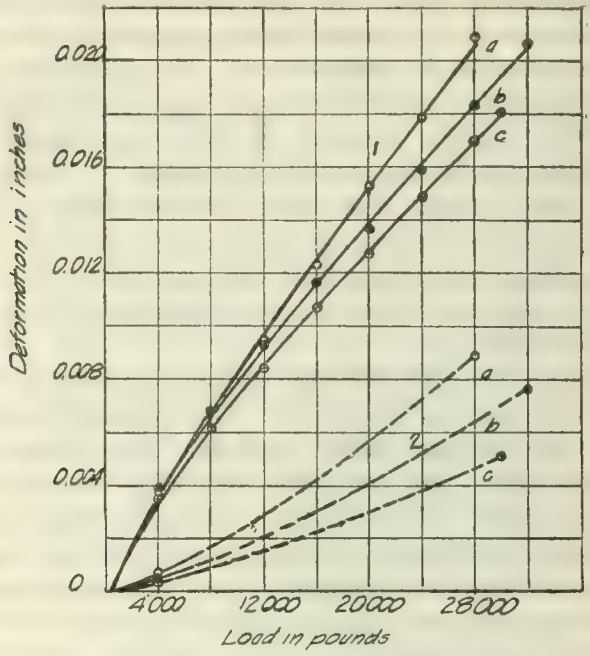

FIG. 23.—*Heavy compression test on 1-inch, 1¼-inch, and 1½-inch balls and races of 0.550-inch, 0.700-inch, and 0.841-inch radius*

Curves in group 1 show total deformation; curves in group 2 show permanent set; curve *a* is the test on the 1-inch ball, curve *b* is on the 1¼-inch ball, curve *c* on the 1½-inch ball

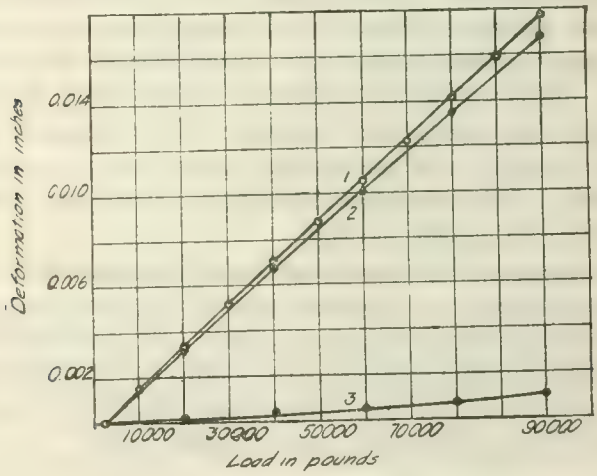

FIG. 24.—*Compression test on 1¼-inch roller between races of 3.5 inch and 4.75 inch radius*

Curve 1 shows total deformation, curve 2 shows elastic deformation, and curve 3 shows the permanent set

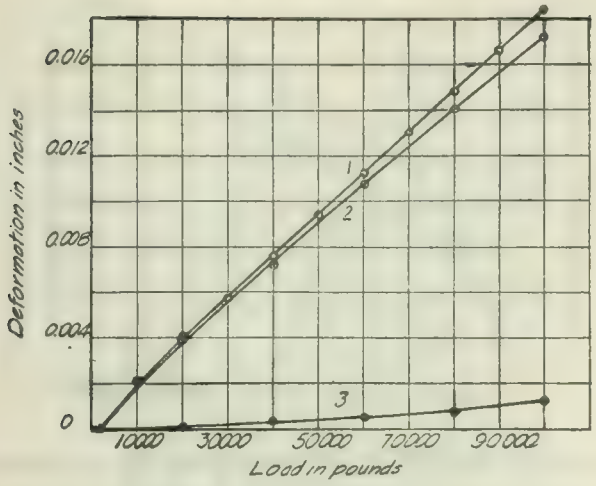

FIG. 25.—*Compression test on 1¼-inch roller between plates*

Curve 1 shows total deformation, curve 2 shows elastic deformation, and curve 3 shows permanent set

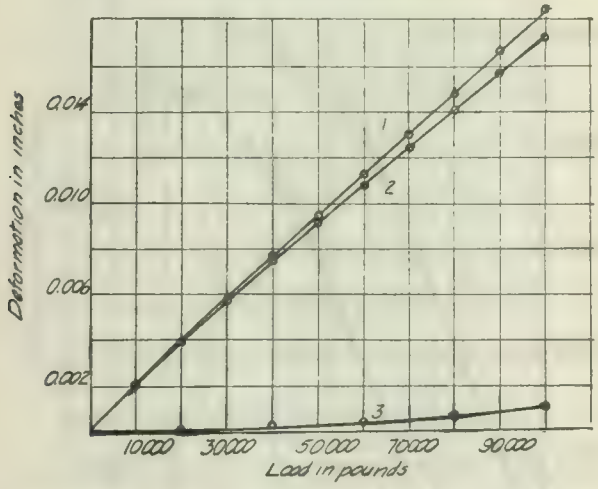

FIG. 26.—*Compression test on 1¼-inch roller between races of 10-inch and 11.25-inch radius*

Curve 1 shows total deformation, curve 2 shows elastic deformation, and curve 3 permanent set

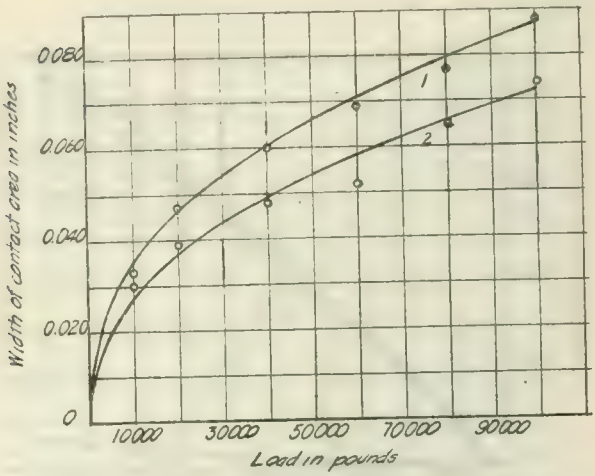

FIG. 27.—*Area of contact of 1¼-inch roller between races of 10 inch and 11.25 inch radius*

Curve 1, outer race; curve 2, inner race

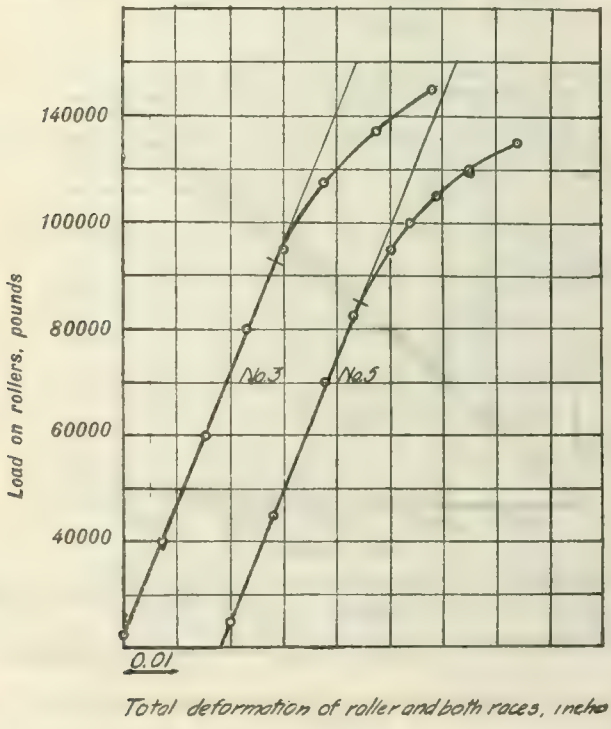

FIG. 28.—*Compression test on 1¼-inch rollers No. 3 and No. 5 with radius of inner race 3.5 inches*

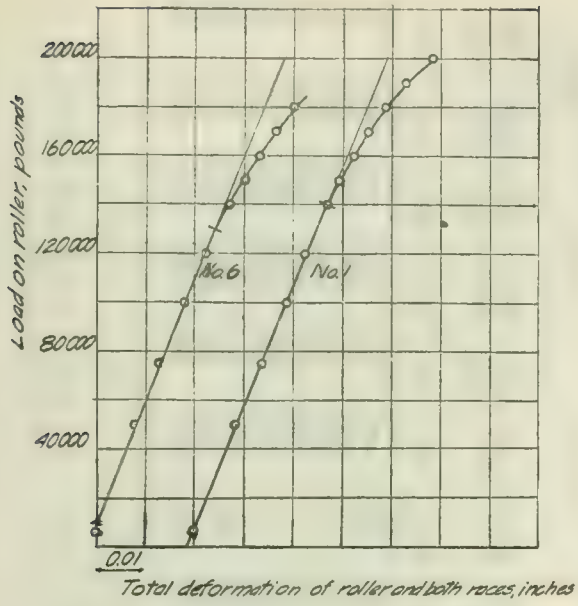

FIG. 29.—*Compression tests on 1¼-inch rollers No. 6 and No. 1 with radius of inner race 10 inches*

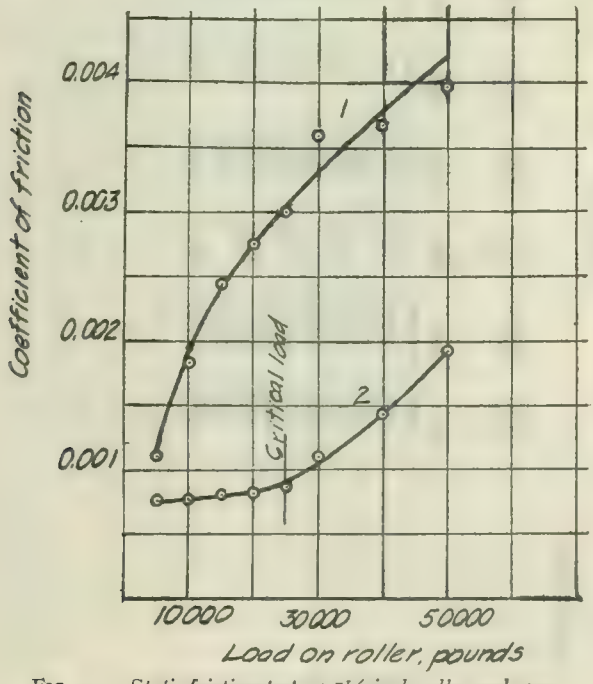

FIG. 30.—*Static friction test on 1¼-inch roller and races*

Curve 1, $r = 3.5$ inches; curve 2, $r = 10.0$ inches

TABLE 15.—Compression Test of Roller (1.25 Inches Diameter), Radius of Inner Race=3.5 Inches

Load in pounds	Total deformation		Permanent set of roller inner and outer race		Width of contact area						Contact area			
					Inner race			Outer race			Inner race		Outer race	
	Observed value	Graph value	Observed value	Graph value	Observed value	Graph value	Hertz value for solid roller	Observed value	Graph value	Hertz value for solid roller	Flexible roller (actual)	Solid roller (theoretical)	Flexible roller (actual)	Solid roller (theoretical)
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch ^{2}	Inch ^{2}	Inch ^{2}	Inch ^{2}
10 000	0.00198	0.00198	.....	0.00006	0.033	0.027	0.020	0.035	0.036	0.022	0.1215	0.0900	0.1620	0.0990
20 000	.00382	.00387	0.00016	.00012	.041	.038	.028	.051	.049	.030	.1710	.1260	.2206	.1350
30 000	.00570	.00573	.....	.00019	.....	.045	.034	.....	.058	.037	.2026	.1530	.2608	.1665
40 000	.00751	.00751	.00031	.00026	.047	.049	.040	.065	.065	.043	.2206	.1800	.2923	.1934
50 000	.00930	.00930	.....	.00034	.....	.054	.044	.....	.070	.049	.2430	.1980	.3148	.2205
60 000	.01104	.01110	.00049	.00046	.053	.057	.049	.075	.075	.053	.2566	.2205	.3373	.2386
70 000	.01280	.01288	.....	.00060	.....	.061	.052	.....	.079	.057	.2745	.2340	.3555	.2567
80 000	.01467	.01467	.00073	.00073	.063	.064	.056	.083	.083	.062	.2880	.2520	.3734	.2790
90 000	.01648	.01648	.....	.00088	.....	.068	.059	.....	.087	.065	.3060	.2654	.3914	.2924
100 000	.01830	.01830	.00100	.00105	.073	.071	.063	.091	.091	.068	.3193	.2835	.4093	.3060

TABLE 16.--Compression Test of Roller (1.25 Inches Diameter), Radius of Race=10 Inches

Load in pounds	Total deformation		Permanent set of roller inner and outer race		Width of contact area						Contact area			
					Inner race			Outer race			Inner race		Outer race	
	Observed value	Graph value	Observed value	Graph value	Observed value	Graph value	Hertz value for solid roller	Observed value	Graph value	Hertz value for solid roller	Flexible roller (actual)	Solid roller (theoretical)	Flexible roller (actual)	Solid roller (theoretical)
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch ^{2}	Inch ^{2}	Inch ^{2}	Inch ^{2}
10 000	0.00208	0.00210	.....	0.00003	0.030	0.027	0.020	0.033	0.034	0.022	0.1216	0.0900	0.1530	0.0990
20 000	.00394	.00403	0.00010	.00008	.038	.037	.028	.047	.046	.030	.1666	.1262	.2070	.1350
30 000	.00588	.00590	.....	.00013	.....	.044	.034	.....	.054	.036	.1980	.1530	.2430	.1620
40 000	.00770	.00770	.00034	.00020	.048	.049	.040	.060	.060	.042	.2207	.1800	.2700	.1890
50 000	.00949	.00948	.....	.00030	.....	.054	.045	.....	.066	.048	.2430	.2027	.2970	.2160
60 000	.01128	.01125	.00047	.00043	.052	.058	.050	.069	.071	.052	.2610	.2250	.3195	.2340
70 000	.01301	.01303	.....	.00057	.....	.062	.053	.....	.075	.057	.2790	.2385	.3375	.2566
80 000	.01482	.01484	.00075	.00072	.065	.066	.057	.077	.080	.061	.2970	.2566	.3600	.2747
90 000	.01670	.01665	.....	.00088	.....	.069	.060	.....	.083	.063	.3105	.2700	.3734	.2835
100 000	.01851	.01840	.00110	.00110	.074	.072	.064	.088	.087	.067	.3240	.2880	.3915	.3015

TABLE 17.—Compression Test of Roller (1.25 Inches Diameter), Between Two Plates

Load in pounds	Total deformation		Permanent set of roller and plates		Width of contact area			Contact area	
	Observed value	Graph value	Observed value	Graph value	Observed value	Graph value	Hertz value for solid roller	Flexible roller (actual)	Solid roller (theoretical)
	Inch	Inch	Inch	Inch	Inch	Inch	Inch	Inch²	Inch²
5 000	.....	.....	.....	.....	0.022	0.021	0.015	0.0945	0.0675
10 000	0.00243	0.00246	.....	0.00007	.030	.029	.021	.1305	.0945
20 000	.00444	.00434	0.00008	.00013	.039	.039	.029	.1755	.1305
30 000	.00617	.00620	.....	.00020	.046	.046	.036	.2070	.1620
40 000	.00800	.00803	.00035	.00029	.051	.051	.041	.2296	.1845
50 000	.00979	.00983	.....	.00040	.055	.055	.046	.2474	.2070
60 000	.01162	.01164	.00049	.00053	.059	.059	.051	.2655	.2295
70 000	.01345	.01345	.....	.00068	.063	.063	.055	.2835	.2474
80 000	.01522	.01523	.00079	.00085	.067	.067	.059	.3015	.2653
90 000	.01701	.01702	.....	.00103	.071	.071	.062	.3193	.2787
100 000	.01892	.01880	.00128	.00125	.076	.075	.065	.3372	.2924

TABLE 18.—Compression Test of Roller (1.25 Inches Diameter)

Specimen (roller) No.	Radius of inner race 3.5 inches		Radius of inner race 10.0 inches	
	Proportional limit	Ultimate strength	Proportional limit	Ultimate strength
	Pounds	Pounds	Pounds	Pounds
3	143 000	211 000	.....	.....
5	130 000	199 000	.....	.....
6	.....	.....	130 000	184 000
1	.....	.....	140 000	206 000

WASHINGTON, March 21, 1921.

DEPARTMENT OF COMMERCE

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S. W. STRATTON, DIRECTOR

No. 202

RESULTS OF A SURVEY OF ELEVATOR INTERLOCKS AND AN ANALYSIS OF ELEVATOR ACCIDENT STATISTICS

BY

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Bureau of Standards

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RESULTS OF A SURVEY OF ELEVATOR INTERLOCKS AND AN ANALYSIS OF ELEVATOR ACCIDENT STATISTICS

By C. E. Oakes and J. A. Dickinson

ABSTRACT

This report gives the results of a field survey of several thousand elevator landings equipped with various types of mechanical and electromechanical interlocks and contact devices. The survey was conducted in connection with the preparation of an elevator safety code, in which work the Bureau of Standards engineers have cooperated with engineers of the American Society of Mechanical Engineers. The elevators are classified: A, Elevators in buildings having heavy service and where maintenance service is provided; B, elevators located in buildings where the service is heavy but without maintenance; and C, elevators on which the service is light and for which no maintenance service is provided. The statistics show that 73.8 per cent of all fatal accidents might be prevented by well-designed interlocks.

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I. INTRODUCTION

During the European War the Bureau of Standards cooperated with the Federal safety engineers in preparing elevator rules, intended primarily for the use of Government navy yards, arse-

nals, and explosive plants. The code was based on existing regulations as prepared by various States and municipalities, together with suggestions from elevator manufacturers and safety engineers. This work was completed in the spring of 1918. Following the completion of this code the Bureau was requested to revise the Federal code to render it suitable for general application.

The American Society of Mechanical Engineers published an elevator code in 1917, which was prepared by the subcommittee on the protection of industrial workers. A complete revision of this code was undertaken a few months after its publication. In order to avoid duplication of effort, the Bureau in 1918, at the invitation of the American Society of Mechanical Engineers, began to cooperate in the further development of this code. The Bureau placed the data it had collected at the disposal of the American Society of Mechanical Engineers' committee and later, in 1919, abandoned all idea of separate development and publication of an elevator safety code, but continued to cooperate with the A. S. M. E. committee with the idea of developing a single national code.

Early in the work it became evident that the point of greatest controversy was the question of proper hoistway-door protection. The Bureau of Standards decided to make a field survey of the various types of interlocks and contacts in use and did so in 1918 and 1919. The results were submitted to the committee for its use in drafting the portion of the code dealing with such devices.

This paper gives the final results of the survey and a collection of statistics from other sources which have been classified and arranged to show the relation of accidents to hoistway-door protection.

II. REPORT ON A FIELD SURVEY OF ELEVATOR INTERLOCKS AND CONTACTS

In the preparation of a safety code of any kind the need for reliable data upon which to base the various requirements is self-evident.

There are certain hazards which are obviously apparent. It may be, however, that hazards of a far graver nature exist which are not apparent to a person making a casual survey of the industry. Ofttimes these less apparent hazards prove to be more serious than those readily apparent. A careful study of accident statistics taken from as large a number of sources as possi-

ble will indicate in a general way the particular parts or types of machines responsible for the more serious accidents.

A summary of the accidents given in the appended tables will show that by far the larger portion of elevator accidents occur at the hoistway door. The question, therefore, of the best method of reducing this hazard is one to which considerable study has been given.

1. HOISTWAY-DOOR PROTECTIVE DEVICES

All forms of hoistway-door protective devices may be broadly classified in three divisions, according to the method of locking the car and the hoistway door.

THE MECHANICAL TYPE.—This type of door safety depends on mechanical action to lock the car-control mechanism while the door is open and to hold the door in the locked position when the car is not at the landing. The functions of this device are generally as follows: (*a*) Release of hoistway door, (*b*) locking control mechanism, (*c*) release of control mechanism, and (*d*) locking of hoistway door.

In most types the locking of the hoistway door is accomplished by holding locked, either by means of a very substantial latch or a properly shaped slot in a rotating member, a bar fastened rigidly to the hoistway door. Generally, the door-locking mechanism is substantial in construction and will stand a large amount of wear without loss of function. Mechanisms which do not depend entirely on spring action to hold the door locked are preferable.

The methods of interlocking the car and door are varied. A variety of methods are necessary, owing to the large number of different types of control mechanism used on different makes and types of elevators. Two popular styles in use are (*a*) a slotted flap which upon the opening of the landing door drops from a vertical to a horizontal position, the slot engaging and holding inoperative the car-control lever in the neutral position; (*b*) a device in which an arc of a circle properly slotted is fastened to the control lever. A bolt or rod engages in this slot, thus preventing the motion of the operating lever when the lever is in the neutral position and the car door open. As a rule these devices are satisfactory. Occasionally, however, installations are found in which there is too much lost motion, and in certain hydraulic installations it is possible to start the car very slowly, due to the slight motion allowed on the control lever when the

car is in the "locked" position. This objection does not ordinarily hold true in case of electrically controlled elevators, as the operating lever must move through an appreciable arc before the operating circuit is completed, although in controllers with a number of contact steps the first contact may be so close to the "stop" position that cases have been noted in which the car may be moved slowly with the interlocking mechanism in the "locked" position. As installed at the present time, a great many of these devices have accessible metal parts in the car, the removal or blocking of which may render the device inoperative. Such parts as are located within the car should be inclosed.

One of the features generally incorporated in this type of lock is some form of limiting device which will prevent the opening of the hoistway door except when the car is within 3 or 4 inches of the landing at which the stop is to be made.

The inclusion of a landing range in an interlock device is not a necessary adjunct to the interlocking function. It is a mechanical detail which is inherent in certain types of devices where interlocking is secured by mechanical means. The absence of a landing range detracts in no way from the proper functioning of a well-designed interlock. The use of a device of this kind is intended to lessen the number of accidents due to persons tripping on entering or leaving the car. Such accidents are, however, relatively unimportant, the minor nature of these accidents being emphasized by the fact that statistics concerning them are not generally recorded separately.

At the request of the Bureau of Standards a list of tripping accidents was prepared by one of the largest casualty insurance companies, the list including all tripping accidents occurring on elevators insured by them during the year 1919. An analysis of this list shows that more tripping accidents occurred when the car platform was within 3 inches of the landing level than occurred when the car platform was more than 3 inches away. In other words, a 3-inch stopping range would not have prevented the majority of these accidents. The consideration of a more or less parallel case may be of interest: A person frequently trips over a projection in a sidewalk where the difference of elevation is slight, but almost never trips over a curbstone where the difference of elevation is considerable. It appears that threshold illumination is an important consideration in preventing tripping accidents and should be given careful attention, if such accidents are to be avoided.

Leveling devices, both automatic and manually controlled, are now on the market and can be used on either passenger or freight elevators. These devices bring the car flush with the landing, and in the case of the automatic device maintain it there during loading and unloading and are doubtless the most effective devices available for decreasing accidents from tripping.

ELECTROMECHANICAL.—The second broad division into which door safety devices may be placed is the electromechanical type, in which the door is mechanically held closed but in which the car is interlocked by means of some form of electrical control. This may be accomplished either by interrupting the master or control circuit or by running an independent circuit which will interrupt the operating current on the machine control board or, in the case of hydraulic installations, by using a current to actuate a solenoid which in turn mechanically locks the car-control lever or holds closed a valve in the supply line, thus preventing motion of the car.

This type of device (electromechanical) may or may not limit the stopping of a car to within a few inches of the landing platform before the door can be opened. Those devices in which the electric contact is placed under the car must necessarily be provided with a stopping range limit, and their operation prevents the opening of the hoistway door unless the car is within a predetermined distance of the landing floor level, while those depending on switches placed in the hoistway can generally be opened without respect to the relative position of the car and hoistway landing platform.

The door-locking mechanisms used in devices of this kind are varied. Those with the undercar contact generally use the same mechanism as is employed in the purely mechanical type, while those with a hoistway contact generally make use of a bar-lock or some form of toggle-joint device. Of the two the latter is preferable, as it does not depend on springs to hold the door locked.

ELECTRIC CONTACT DEVICES.—The third class of door safety devices comprises a wide variety of electric hoistway-door contacts. These devices are generally connected in series known as shaft-series system and operate on the car-control circuit, although they may be wired so that each door is a separate unit, this wiring being known as a door-unit system. Those in which the switch is actuated by the door itself (there are a large variety of this type) give considerable trouble, due to lack of alignment of the door and switch and to vibration and the impact of the door.

The difference between a hoistway-door contact and an electromechanical interlock should be clearly kept in mind. A contact device holds the car inoperative when the door opposite which the car is standing is open. An electromechanical interlock is a device which holds the car inoperative when the door opposite which the car is standing is open or unlocked and in addition holds locked the hoistway door when the car is not at the landing. This door-locking function is accomplished by a mechanism independent of, and separate from, the ordinary door latch.

In other words, a contact is a device which can only prevent the motion of the car when a door is open. It can not and does not hold locked the door when the car is not at the landing. The ordinary hoistway-door latch must be used as the door-locking device.

The electromechanical device has a bar or other form of lock incorporated in it and so arranged and interconnected that the door must be closed and locked before the car can be started.

In order to obtain information regarding the effectiveness of the ordinary hoistway-door latch, a survey was made of several hundred hoistway doors. The results of this survey indicate that 30 per cent of the latches were out of order, the principal causes of failure being worn latch parts, worn door hangers, loosening of latch from door, and lack of lubrication. Many latches were so poorly designed that if the door were closed quickly the rebound of the door would occur before the latch engaged. An analysis of elevator-accident statistics shows that about one-third of the accidents occur by persons falling down the shaft. This is in itself an indication that the ordinary latch is unreliable and shows why a contact device can not of itself offer complete hoistway-door protection. In the other two types of device (mechanical and electromechanical) the door is held shut mechanically whether or not the car interlocking feature is working.

2. EMERGENCY RELEASE

Every elevator car equipped with interlocks should be provided with an emergency switch or release device located in the car. If there is a fire, panic, or other emergency, or if the car becomes stalled in the shaft due to failure of the door safety circuit, the operation of the emergency device will permit the elevator to continue in operation. Unless this device is properly designed and installed its use may be subject to much abuse.

An undesirable feature of an emergency switch so designed that the switch can be turned "on" or "off" is that, in event of failure of any contact, the throwing of an emergency switch leaves the entire system with no protection other than that offered by the ordinary door latch, which is notoriously unreliable. The emergency release generally consists of a two-pole push or snap switch sometimes placed in a case under a glass cover, often without even this protection, and when once thrown is quite likely to be left in the "emergency" position for weeks or months. Except where required by law, the spring-type switch, which must be held in the emergency position by the operator, is seldom seen, although it is obviously the most desirable type. The emergency-release device for mechanical locks which remains in the emergency-operating position without being held there by the operator is equally undesirable.

3. FIELD SURVEY

In order to obtain reliable information regarding door protective devices, a large number of passenger elevators were inspected by representatives of the Bureau of Standards and thousands of doorways and landings examined. These inspections covered a considerable range of territory, and territories in which the legal requirements for the protection of entrance doors to elevators differ widely. The following statement gives the various cities and States in which data were secured and the legal requirements in each place:

	Requirements
Philadelphia	Mechanical or electromechanical interlocks of the undercar contact type.
Pennsylvania (outside of cities of first or second class)	Mechanical or electromechanical interlocks and car-door contacts on cars driven by electricity.
New Jersey	Mechanical or electromechanical.
New York City	No interlock requirements.

4. DIVISION OF BUILDINGS INTO CLASSES ACCORDING TO CONDITIONS OF SERVICE AND MAINTENANCE

For the convenience in analyzing the performance under actual operating conditions of any given interlock device or group of such devices it is desirable to divide the elevators which they protect into three general classes, the classification being naturally somewhat elastic. The two factors which govern this classi-

fication are the amount of duty rendered and the maintenance service they receive. The grouping adopted is as follows:

Class A.—Those in which the volume of travel is heavy and for which there is a special elevator maintenance and inspection service. Into this class will fall: (*a*) Large office buildings, (*b*) first-class hotels, (*c*) high-grade apartment houses, (*d*) large railroad terminals, (*e*) municipal and State buildings in cities, and (*f*) large department stores. This class is generally limited to the larger cities.

Class B.—Buildings in which the travel is heavy, but in which no regular maintenance or inspection service is provided. In this are included the following types of buildings: (*a*) Small office buildings in large cities, (*b*) large office buildings in small cities, (*c*) loft and manufacturing buildings, (*d*) small hotels in cities, (*e*) medium-size apartment houses, (*f*) small department stores in cities, and (*g*) other retail stores in cities.

Class C.—Buildings that have light service and receive but little care and attention. In this class will fall: (*a*) Small office buildings in towns or small cities, (*b*) storage or warehouse buildings, (*c*) small hotels, (*d*) manufacturing plants where the upper floors are used largely for storage, and (*e*) very small shops and retail stores (such as furniture stores) where most of the selling is done on the ground floor.

5. DISCUSSION OF CONDITIONS UNDER WHICH DEVICES MUST OPERATE

In Class A buildings the vital consideration is speed of service. In most installations of this class there are several load peaks, and at such times every car is run at capacity. A loss of a half minute a trip per elevator would mean a serious curtailment of service. Any device that is installed must not reduce the operating speed to any marked degree. Further, it must be provided with some form of emergency release, so that the failure of the device or of any part of it will not tie up the car during the rush period.

The survey showed that there is generally very little operating trouble due to failure of door-safety equipment in Class A buildings, because ample maintenance service is provided. As a rule, the elevator maintenance crew take pride in the condition of the equipment under its care and keep parts properly adjusted and in repair.

In Class B buildings the vital consideration is simplicity and ruggedness, as the service in this class of buildings is severe and

the maintenance and attention which elevator parts and devices will receive is slight. Lack of alignment on the part of the elevator or of the hoistway door may cause excessive wear of the interlock or contact, causing rapid deterioration, which results ultimately in failure to operate.

As a usual thing the only attention which a Class B elevator receives is an occasional "going over" by some mechanic or handy man for the purpose of lubrication. Even this perfunctory oiling is done at odd times and at very irregular intervals. It is very seldom that the elevators in a Class B building are in charge of a starter, and it is only in very exceptional cases that the operators are given any definite instructions as to the operation of the car; in other words, operating rules are usually unknown.

This class (Class B) will contain by far the largest part of the elevators in almost any city or large town. For this reason the behavior of the various door-protective devices in Class B buildings is a matter of vital interest. For this class of service the most simple and rugged type of mechanical interlock or electromechanical device is desirable. Almost invariably electric door contacts seem to give trouble, under the combination of heavy duty and little or no attention.

For elevators in Class C buildings the requirements which are to be met are not so severe. Any device that will stand up under occasional use, but with almost entire lack of attention, will probably give satisfaction. Most mechanical devices give fairly good service, although in some cases which were noted they have worn rapidly owing to lack of lubrication. Electromechanical and contact devices give fair service provided they are used often enough to prevent the contacts from oxidizing and corroding. This class is probably smaller than either Classes A or B, and is confined very generally to smaller cities and towns. Frequently in Class C installations may be found various types of makeshift devices, some of which fail to accomplish the purpose for which they were intended.

6. TABULATION OF RESULTS OF SURVEY

The data regarding interlocks and electric contacts given on the following pages were obtained by examining substantially all of the elevators in many of the smaller cities and towns of Pennsylvania and New Jersey and all of the elevators in representative blocks in the larger cities. In uptown New York the blocks were

between Fifth Avenue and Broadway and Thirty-sixth and Forty-fifth Streets. In downtown New York both sides of Broadway were covered for a number of blocks. This was done in an effort to observe an average condition, and it is believed that the data given here are typical of the various towns or sections in which inspections were made.

Summary of Conditions of Interlocking Devices in Pennsylvania (Exclusive of First-Class Cities)

GENERAL ANALYSIS

Elevators inspected	79
Landings inspected	513
Cars not provided with car-gate contacts (required by Pennsylvania law)	20
Per cent not up to requirements of Pennsylvania law	25.2
Landings provided with interlocks	513
Landings provided with interlocks found to be inoperative	47
Per cent of landings with interlocks inoperative	9.2

ANALYSIS BY TYPE OF DEVICE

Landings provided with mechanical interlocks	244
Landings with mechanical interlocks inoperative	26
Per cent of landings with mechanical interlocks inoperative	10.6
Landings provided with electromechanical interlocks	269
Landings provided with electromechanical interlocks found to be inoperative	21
Per cent electromechanical interlocks found to be inoperative	7.8

ANALYSIS BY CLASS OF BUILDINGS IN WHICH ELEVATORS ARE LOCATED

CLASS A

Landings in this class	149
Landings with interlocks inoperative	10
Per cent of landings with interlocks inoperative	6.7

CLASS B

Landings in this class	313
Landings with interlocks inoperative	36
Per cent of landings with interlocks inoperative	11.5

CLASS C

Landings in this class	51
Landings with interlocks inoperative	1
Per cent of landings with interlocks inoperative	2.0

Summary of Conditions of Interlocking Devices in New Jersey

GENERAL ANALYSIS

Elevators inspected	189
Landings inspected	1466
Landings with no hoistway-door protection	227
Per cent of landings with no hoistway-door protection	15.5
Landings not provided with mechanical or electromechanical interlocks (as required by New Jersey law)	559

Per cent of landings not provided with mechanical or electromechanical interlocks (as required by New Jersey law)	44.8
Landings provided with interlocks or contacts	1239
Landings with interlocks or contacts inoperative	73
Per cent of landings with interlocks or contacts inoperative	5.2

ANALYSIS BY TYPE OF DEVICE

Landings provided with mechanical interlocks	243
Landings provided with mechanical interlocks found to be inoperative	29
Per cent of mechanical interlocks found to be inoperative	11.9
Landings provided with electric contacts [1]	329
Landings provided with electric contacts found to be inoperative [1]	44
Per cent of electric contacts found to be inoperative [1]	13.3
Landings provided with electromechanical interlocks [2]	667
Landings provided with electromechanical interlocks found to be inoperative [2]	0
Per cent of electromechanical interlocks found to be inoperative [2]	0

ANALYSIS BY CLASS OF BUILDING IN WHICH ELEVATORS ARE LOCATED

CLASS A

Landings in this class	845
Landings with interlocks or contacts inoperative	19
Per cent of landings with interlocks or contacts inoperative	2.2

CLASS B

Landings in this class	350
Landings with interlocks or contacts inoperative	53
Per cent of landings with interlocks or contacts inoperative	15.1

CLASS C

Landings in this class	44
Landings with interlocks or contacts inoperative	1
Per cent of landings with interlocks or contacts inoperative	2.3

Summary of Conditions of Interlocking Devices in Philadelphia

[NOTE.—Municipal regulations require the use of mechanical or undercar contact electromechanical interlocks.]

GENERAL ANALYSIS

Elevators inspected	49
Landings inspected	503
Landings provided with interlocks	503
Landings with interlocks inoperative	66
Per cent of landings with interlocks inoperative	13.0

ANALYSIS BY CLASS OF BUILDING IN WHICH ELEVATORS ARE LOCATED

CLASS A

Landings in this class	404
Landings with interlocks inoperative	9
Per cent of landings with interlocks inoperative	2.2

[1] While contact devices without door-interlocking mechanisms are not permitted under the New Jersey law, a number of such devices were found. Some of these devices were installed prior to the enactment of the law and will, no doubt, eventually be changed to comply with the legal requirements.

[2] Seventy-five per cent of the electromechanical devices were found in one "Class A" group, where the maintenance service was excellent.

CLASS B

Landings in this class	99
Landings with interlocks inoperative	57
Landings with interlocks inoperative (final)	28
Per cent of landings with interlocks inoperative	56.0
Per cent of landings with interlocks inoperative (final)	28.0

Summary of Conditions of Interlocking Devices in Uptown New York City [4]

GENERAL ANALYSIS

Elevators inspected	294
Landings inspected	3,043
Landings provided with interlocks or contacts	229
Per cent of landings provided with interlocks or contacts	7.5
Landings with interlocks or contacts inoperative	50
Per cent of landings with interlocks or contacts inoperative	21.8

ANALYSIS BY TYPE OF DEVICE

Landings provided with electric contacts	190
Landings with inoperative electric contacts	37
Per cent of landings with inoperative electric contacts	19.2
Landings provided with electromechanical interlocks	39
Landings provided with electromechanical interlocks inoperative	13
Per cent of landings with electromechanical interlocks inoperative	33.3

DATA ON CAR-GATE CONTACTS

Cars provided with car-gate contacts	22
Car-gate contacts in operating condition	10
Per cent of car-gate contacts in operating condition	45.5
Car-gate contacts tied up or plugged	7
Per cent of car-gate contacts tied up or plugged	31.8
Car-gate contacts broken or defective	5
Per cent of car-gate contacts broken or defective	22.7

Summary of Conditions of Interlocking Devices in Downtown New York City [5]

GENERAL ANALYSIS

Elevators inspected	241
Landings inspected	5,304
Landings provided with interlocks or contacts	1,926
Per cent of landings provided with interlocks or contacts	36.4
Landings with interlocks or contacts inoperative	2
Per cent of landings with interlocks or contacts inoperative	0.10

ANALYSIS BY TYPE OF DEVICE

Landings provided with mechanical interlocks	892
Landings with mechanical interlocks inoperative	1
Per cent of landings with mechanical interlocks inoperative	0.10
Landings provided with electromechanical interlocks	1,014
Landings with electromechanical interlocks inoperative	1

[3] As 29 of the 57 defective landings were in a single group of buildings which had evidently been overlooked by the inspector, this estate was omitted from the final statistics as not being representative.

[4] Only one mechanical interlock was found in the particular uptown section inspected, so that statistics on this type are not given. Buildings equipped with interlocks or contacts in this tabulation may be regarded with very few exceptions as Class B.

[5] All of these downtown buildings provided with interlocks or contacts in this tabulation are Class A buildings.

Per cent of landings with electromechanical interlocks inoperative	0. 10
Landings provided with electric contacts	20
Landings provided with electric contacts inoperative	0

Summary of an Inspection to Determine the Effectiveness of Emergency Releases, Uptown New York City [6]

ELECTROMECHANICAL DEVICES

Elevators inspected	43
Landings inspected	504
Landings working satisfactorily	225
Per cent of landings working satisfactorily	44. 6
Landings where elevators were running with emergency switch thrown; no electrical or mechanical trouble	148
Per cent of landings unprotected, due to emergency switch being thrown	29. 4
Landings where elevators were running with emergency switch thrown or system disconnected, due to electrical or mechanical breakdown	131
Per cent of landings unprotected, due to electrical or mechanical breakdown	26. 0

This survey was made to determine to what extent the emergency release is used as a means of defeating the purpose of door-protective devices.

The emergency releases were all of the two-pole, push-button type. Most of them were without a cover glass or even a means of holding a cover glass. The results of this survey show that an emergency release of this type will be misused. The only type which should be permitted is that in which the operator is required to hold the emergency release in the emergency operating position.

Practically all of the devices inspected on this survey were poorly designed and cheaply made, which may in part account for the large number of defective landings noted.

7. DISCUSSION OF SURVEY

Relation of Law Enforcement to Elevator Safety.— There is a marked difference between the condition of elevators in a State or city where the law or code relating to elevators is strictly enforced and a State or city where the law does not have the backing of an active and thorough administrative organization. In many parts of one State the law calling for interlocks did not seem to be generally known, and even in the larger cities no particular efforts had been made to enforce it. In many cases it has been through the notification given the building owners by insurance inspectors that the devices have been installed. By way of contrast, however, in another State there was not a single elevator examined that was not equipped with the required inter-

[6] Nearly all of these are Class B buildings.

lock, although in a few electric-elevator installations the additional car-door contact which was required by law was lacking.

Perhaps the most badly needed addition to present State rules and city ordinances is a set of simple, brief, and concise operating rules to be posted in every elevator under the jurisdiction of the particular State or city. Certain cities have already provided for the licensing of car operators. The passing of an examination is necessary before the applicant is granted a license. In this way it is possible to discourage the operation of elevators by the ignorant or uninstructed. Unfortunately, during the war it was deemed necessary to waive this ruling in certain cases, and the strict enforcement of this rule has not yet been insisted upon.

Performance of Interlocks in Various Classes of Buildings.—In regard to the performance of interlocking devices, there seems to be but little difference between the mechanical and electromechanical interlocks. Many installations, both mechanical and electromechanical, were met with in Philadelphia that had been giving satisfactory service for periods of 15 years and upward, with a cost for repairs and renewals not exceeding $1 per landing for the entire period, while numerous installations met with in the smaller cities of Pennsylvania were found to be in good condition after 10 years or more of service.

As might be expected as the result of the combination of hard service with little or no maintenance, the Class B installations in every case have made the poorest showing, but even here the percentage of installations in working order indicates that the protection afforded by such devices is decidedly worth while.

For Class A installations the average percentage of devices in working order was 98. When the fact that an interlock may be called on to function several hundred times a day is taken into consideration, this percentage must be considered remarkable.

Class C installations were generally in fairly good condition, due mainly to light service requirements.

Interlock Installations in Large Office Buildings.—The results of the inspection in the down-town section of New York City brings forth two points of particular interest.

First, the relatively large number of interlock installations. While interlocks are not required by the city ordinances, over one-third of the landings inspected had been so safeguarded.

Second, the remarkably low number (two) of defective landings found despite the severe duty imposed on the elevators in the

large office buildings down town. Excellent maintenance service and the carefully designed and well-built interlocks which predominate in this section are, no doubt, largely responsible for this fine showing.

Suggestions for Improvement in Design.—From a general survey of results obtained it would seem that there is room for considerable improvement in the types of interlocking devices. Of the hundreds of elevators inspected none were equipped with a mechanical interlock in which the four vital functions of unlocking hoistway door, locking car mechanism, unlocking car mechanism, and locking the door are performed by means of positive mechanical motion. In other words, at least one, and generally more, of these functions are accomplished by means of springs or the force of gravity, or a combination of the two. An interlock in which all four of these functions are performed without the aid of a spring or the action of gravity should be not particularly difficult of design. Rules to be drafted should, however, take cognizance of present well-designed apparatus and should be so worded as to not hinder improvement in design or new designs.

Door contacts, while usually of ample current-carrying capacity, are seldom rugged enough to stand up under the impact and vibration of the hoistway door, where they are actuated by the impact of the door itself. Another characteristic feature about them is that in nearly every case the making and breaking of the current is dependent directly upon and is accomplished at the same speed as the opening and closing of the door. Many such devices, if the door fails to latch, are left in but slight contact and may produce an arc which will rapidly destroy the contact surfaces. The use of a sequence coil or a quick make-and-break switch will do much to eliminate such arcing and pitting.

There are several well-designed devices with switches that are not subject to door impact and which give long and satisfactory service when made a part of a substantial interlock mechanism. Well-designed mechanical locks are also available. A substantial door-locking mechanism is necessary whether the method of interlocking the car is electrical or mechanical. The door-locking function should be so interconnected with the car interlock that the car can not be started until the door is locked in the closed position.

There is considerable need for some sound engineering work in connection with the proper design of interlocks. Many of the devices on the market were designed without due regard for the stresses set up by door impact, lack of alignment, and abuse of equipment at the hands of careless operators. Such devices are generally sold at a low price, the manufacturer depending upon this low price to obtain business. In many cases such devices fail after a short period of service and are, in the end, more expensive than a well-designed and carefully installed outfit, the initial cost of which would have been greater.

III. SUMMARY OF ADVANTAGES AND DISADVANTAGES OF VARIOUS TYPES OF DEVICES

The following tabular arrangements set forth the advantages and disadvantages of the various types of elevator door safety devices and the characteristic methods of failure of the various types of devices for the various classes of service:

Comparison of Functions and Maintenance of Various Types of Elevator Door Devices

Mechanical	Electromechanical		Electric contacts
	Undercar-contact type	Hoistway-contact type	
Locks door and prevents operation of car if door is unlatched or unlocked.	Locks door and prevents operation of car if door is unlatched or unlocked.	Locks door and prevents operation of car if door is unlatched or unlocked.	Does not lock door and does not prevent operation of car if door is unlatched.
Interlocks car when door is open	Interlocks car when door is open	Interlocks car when door is open	Interlocks car when door is open.
Slows down service slightly	Slows down service slightly	Slows down service very slightly	Slows down service very slightly.
Usually designed to prevent opening of shaftway door with key unless car is at the landing.	Usually designed to prevent opening of shaftway door with key unless car is at the landing.	May or may not prevent opening of door with key unless car is at the landing, depending on design.	Door may be opened with key when car is not at landing.
Failure of mechanical parts on one door does not affect protection of other doors.	Failure of electric contact will eliminate interlocking of car, but will not affect the mechanical locking of shaftway doors (partial protection until repairs are made).	Failure of electric contact will eliminate interlocking of car, but will not affect the mechanical locking of shaftway doors (partial protection until repairs are made).	Failure of device at one door necessitates running the entire door system without the interlocking protection until repairs are made. Doors may or may not be latched, depending on condition of door latch.
Prevents opening of door by operator unless car is within a few inches of the landing.	Prevents opening of door by operator unless car is within a few inches of the landing.	Door can be opened when operator is in position to reach door-opening device.	Door can be opened when operator is in position to reach door-opening device.
MAINTENANCE AND OPERATION			
Renewals infrequent	Renewals not frequent	Renewals not frequent	Renewals frequent.
Necessary to use emergency release very occasionally.	Necessary to use emergency release very occasionally.	Necessary to use emergency release occasionally if switch is not subject to door impact; much oftener if switch is subject to such forces.	Necessary to use emergency release fairly often if switch is subject to door impact and vibration, otherwise much less often.
Danger of crushing passengers slight, as doors must be closed and locked before car starts.	Danger of crushing passengers slight, as doors must be closed and locked before car starts. Short-circuit, ground, or an emergency switch thrown may set up a very dangerous crushing hazard.	If contact is not made until door is shut, crushing hazard is slight. If contact is made before door reaches full closure, the entering passenger may be caught and crushed by the starting car. Short-circuit, ground, or an emergency switch thrown may set up a very dangerous crushing hazard.	If contact is not made until door is shut, crushing hazard is slight. If contact is made before door reaches full closure, the entering passenger may be caught and crushed by the starting car. Short-circuit, ground, or an emergency switch thrown may set up a very dangerous crushing hazard.

Comparison of Service of Hoistway-Door Interlocking Devices for the Various Classes of Buildings

CLASS A

Mechanical	Electromechanical	Electric contact
Parts fail through wear due to heavy service.	Parts fail through wear and arcing due to heavy service.	Parts fail by wear and arcing due to heavy service and where the switch is actuated directly by the door, by mechanical breakdown due to door impact and vibration.

CLASS B

Fail through rapid wear and mechanical breakdown due to lack of proper lubrication and lack of alignment combined with severe conditions of service.	Fail through mechanical or electrical breakdown due to lack of proper lubrication, lack of alignment, dirt and grit on contact surfaces combined with severe service conditions.	Fail through mechanical and electrical breakdown due to dirt and grit on contact surfaces, lack of proper lubrication combined with severe service conditions and where the switch is actuated directly by the door, to breakdown due to impact of door, vibration, lack of alignment between door and switch.

CLASS C

Fail through mechanical wear augmented by lack of proper lubrication.	Fail through mechanical wear or failure of contacts due to oxidation, corrosion, or pitting due to infrequent use and dirt on contact surfaces.	Fail through oxidation, corrosion, and pitting of contact due to infrequent use and dirt on contact surfaces.

IV. ELEVATOR-ACCIDENT STATISTICS

There are attached a number of tables of elevator statistics obtained from various sources. The public accidents have been separated from the industrial accidents where this was possible, and the classification has been made to show the number of accidents by causes.

Accidents have been classified as public, semipublic, or industrial, according to the nature of the occupancy of the building, as follows:

Public accidents	Hotel Apartment house Department store (patrons). Office building
Semipublic accidents	Hospital Garage
Industrial accidents	Factory Mercantile establishment (employees only).

The accidents have been further grouped into shaft-door and nonshaft-door accidents and the percentages computed on this basis for each table. For convenience in analysis the statistics have been divided into groups according to sources, as noted below and shown in the tables following:

Group I. General: Reports of accidents received through clipping bureaus. These cover the entire United States.

Group II. State industrial reports of State industrial commissions. These are taken from records of industrial accidents and do not cover accidents to the general public.

Group III. Municipal statistics from records of coroners of cities or records of elevator inspection departments.

1. GROUP I. GENERAL

TABLE 1.—Elevator Accidents Reported Through Clipping Bureaus by the Public Press of the United States, January, 1913, to July, 1918

Cause	Fatal				Nonfatal			
	Public	Semi-public	Industrial	Not classified	Public	Semi-public	Industrial	Not classified.
Fell into shaft from landing floor or due to motion of the car	150	15	158	.....	121	10	181	.....
Crushed between car and sill of landing floor or frame of door	150	10	168	.....	107	5	138	.....
Accident occurred inside shaft	96	11	150	.....	330	12	436	.....
Cause not given	11	0	12	7	11	0	24	11
Total	407	36	528	7	569	27	779	11

TABLE 2.—Summary from Table 1: Entire United States from Newspaper Clippings, January, 1913, to July, 1918

Classification and cause	Fatal	Nonfatal
Public:	Per cent	Per cent
Nonshaft-door accidents	26.2	60
Shaft-door accidents—		
Fell into shaft	36.9	21.3
Crushed between car and sill or frame of door	36 9	18.7
Total	100	100
Semipublic:		
Nonshaft-door accidents	30.5	44.5
Shaft-door accidents—		
Fell into shaft	41.7	37
Crushed between car and sill or frame of door	27.8	18.5
Total	100	100
Industrial:		
Nonshaft-door accidents	49.6	59
Shaft-door accidents—		
Fell into shaft	37.4	23.2
Crushed between car and sill or frame of door	13	17.8
Total	100	100

2. GROUP II. REPORTS FROM STATE INDUSTRIAL COMMISSIONS OR DEPARTMENTS OF LABOR

TABLE 3.—Elevator Compensable Industrial Accidents, New York State, June 1, 1914, to July 1, 1915

[From data submitted by the New York Industrial Commission. N. O. C.=Not otherwise classified]

NONFATAL ELEVATOR ACCIDENTS

Cause	Number	Per cent	Cause	Number	Per cent
Caught between floor and car	113	19.6	Caught between car and gate	13	2.3
Falls into shaft from floor	63	10.9	Struck by counterweight	11	1.9
Caught between shaft and car	26	4.5	Objects falling down shaft from car	13	2.3
Fall of car	41	7.1	Struck by car in pit	9	1.5
Struck by car (N. O. C.)	36	6.3	Machinery breaking	10	1.7
Gates (N. O. C.)	39	6.7	Load catching between car and shaft	6	1.0
Unknown	33	5.7	Cables unwinding	4	.7
Objects falling into shaft	61	10.0	Struck while on top of car	4	.7
Cables breaking	32	5.6	Total	577	100.0
Caught by machinery	25	4.3			
Caught by cable	25	4.3			
Falls into shaft from car	13	2.3			

TABLE 3.—Elevator Compensable Industrial Accidents, New York State, June 1, 1914, to July I, 1915—Continued

FATAL ELEVATOR ACCIDENTS

Cause.	Number.	Per cent.	Cause.	Number.	Per cent.
Falls into shaft from floor	13	27.7	Unknown	1	2.1
Caught between shaft and car	6	12.8	Machinery breaking	1	2.1
Caught between floor and car	5	10.6	Load catching between car and shaft	1	2.1
Struck by car (N. O. C.)	4	8.6	Cable unwinding	1	2.1
Fall of car	3	6.4	Struck while on top of car	1	2.1
Falls into shaft from car	3	6.4			
Struck by counterweight	3	6.4	Total	47	100.0
Struck by car in pit	3	6.4			
Caught between car and gate	2	4.2			

TABLE 4.—Elevator Accidents in Industries and Mercantile Establishments in the State of Pennsylvania, 1918

[From statistics furnished by the Department of Labor and Industry of Pennsylvania]

	Fatal		Nonfatal				Total	
	Number	Per cent	Serious: Number	Minor: Number	Total		Number	Per cent
					Number	Per cent		
Cable breaking	4	8	27	29	56	6.5	60	6.6
Cable unwinding	1	2	9	7	16	1.7	17	1.9
Cable, caught by	3	6	43	49	92	10.7	95	10.5
Car, caught by	17	34	134	171	305	35.7	322	35.5
Car, struck by	3	6	30	36	66	7.8	69	7.6
Counterweight, struck by	2	4	10	7	17	2.0	19	2.0
Defective equipment	1	2	10	17	27	3.1	28	3.1
Entering or leaving car	1	2	10	35	45	5.3	46	5.1
Fall of car	5	10	35	26	61	7.1	66	7.3
Fall of person	7	14	34	27	61	7.1	68	7.5
Falling objects	1	2	15	16	31	3.6	32	3.5
All others	5	10	31	50	81	9.4	86	9.4
Total	50	100	388	470	858	100.0	908	100.0

TABLE 5.—Wisconsin Industrial Accidents on Elevators

[From published reports of the Wisconsin Industrial Commission, 1915–1917]

Cause	Fatal		Nonfatal		Total	
	Number	Per cent	Number	Per cent	Number	Per cent
Falling from floor down shaft	5	35.6	24	11.7	29	13.2
Platform floor catching person	2	14.3	55	26.8	57	26.0
Caught between cage and side	.....	.....	14	6.9	14	6.4
Struck by objects falling down shaft	1	7.1	14	6.9	15	6.9
Car struck person	1	7.1	14	6.9	15	6.9
Car falling	1	7.1	16	7.8	17	7.7
Car sudden start or stop	.....	.....	4	1.9	4	1.8
Struck by counterweight	1	7.1	4	1.9	5	2.3
Hoisting cable striking person	.....	.....	10	4.9	10	4.6
Hoisting machinery catching person	.....	.....	22	10.7	22	10.1
Platform and gate catching person	2	14.4	18	8.8	20	9.1
Falls from car	1	7.1	6	2.9	7	3.2
Load or other object catching person	.....	.....	4	1.9	4	1.8
Total	14	100.0	205	100.0	219	100.0

TABLE 6.—Elevator Accidents in Industries in Massachusetts (Freight and Passenger, Insured, not Insured, and Common Law Rights)

[From published reports of Massachusetts Industrial Accident Board

Cause	July 1, 1913, to June 30, 1914		July 1, 1914, to June 30, 1915		July 1, 1915, to June 30, 1916		July 1, 1916, to June 30, 1917		Total			
									Fatal		Nonfatal	
	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Number	Per cent	Number	Per cent
Caught in machinery	.....	62	.....	40	.....	30	1	47	1	1.5	179	6.92
Caught between car and shaft	7	50	6	186	7	204	6	289	26	38.8	728	28.1
Caught underneath or on top of car	.....	36	.....	16	4	25	3	18	7	10.5	95	3.67
Falling car	1	38	.....	49	1	44	1	47	3	4.48	179	6.87
Falling down shaft (person)	1	59	3	59	9	56	9	61	22	32.9	235	9.08
Struck by falling object	.....	53	1	55	.....	24	.....	29	1	1.5	161	6.22
Caught by fire hatch or trap	.....	9	.....	6	.....	1	.....	9	.....	.....	25	.96
Miscellaneous	.....	284	3	265	2	213	2	228	7	10.5	990	38.2
Total	9	591	13	676	23	597	22	728	67	100.0	2,592	100.0

NOTE.—The classification used here is not as suitable for grouping the hoistway-door accidents into two general classes as the other tables presented.

TABLE 7.—Analysis of State Elevator Statistics

INDUSTRIAL ELEVATOR ACCIDENTS, NEW YORK STATE, JUNE 1, 1914, TO JULY 1, 1915

Cause	Fatal	Nonfatal
	Per cent	Per cent
Nonshaft-door accidents	51.1	64.9
Shaft-door accidents:		
Fell into shaft	34.1	13.2
Caught between floor and car or gate and car	14.8	21.9
Total	100	100

INDUSTRIAL AND MERCANTILE ELEVATOR ACCIDENTS, PENNSYLVANIA, 1918

Nonshaft-door accidents	50	51.9
Shaft-door accidents:		
Fell into shaft	14	7.1
Caught by car and entering or leaving car	36	41
Total	100	100

INDUSTRIAL ELEVATOR ACCIDENTS, WISCONSIN, 1915–1917

Nonshaft-door accidents	35.7	52.7
Shaft-door accidents:		
Fell into shaft	35.6	11.7
Platform floor catching person; platform and gate catching person	28.7	35.6
Total	100	100

INDUSTRIAL ELEVATOR ACCIDENTS, MASSACHUSETTS, JULY 1, 1913, TO JUNE 30, 1917, INCLUSIVE

Nonshaft-door accidents	28.3	62.82
Shaft-door accidents:		
Fell into shaft	32.9	9.08
Caught between car and shaft	38.8	28.1
Total	100	100

3. GROUP III. MUNICIPAL STATISTICS

TABLE 8.—Philadelphia Elevator Accidents

Type	1914		1915		1916		1917		1918	
	Injured	Killed	Injured	Killed	Injured	Killed	Injured	Killed	Injured	Killed
Passenger:[a]										
Electric		1		1			6	1		
Hydraulic	3	1	3	1	5	3	6	3		4
Freight:										
Electric	7	2	14	2	8	2	5	5	5	2
Hydraulic	6	6	4	1	9	6	5	2	2	3
Belt	17	4	22	4	24	2	24	7	12	12
Hand		1	3				3		2	1
Sling							1	1	1	1
Total	33	16	46	9	46	13	49	19	22	23

[a] Interlocks required by law on passenger elevators.

TABLE 9.—Fatal Accidents on the Elevators of Borough of Manhattan, New York City, 1907 to 1918, Inclusive

[From coroner's reports]

Cause	Number	Per cent
Fell down shaft	292	40.5
Crushed between moving car and sill or frame of shaft door	335	46.5
Miscellaneous (broken cables, falling cars, workmen in the shaft struck by moving car, etc.)	94	13
Total	721	100

TABLE 10.—Fatal Elevator Accidents, Cook County, Ill., 1904 to March, 1916, Inclusive

[From coroner's reports]

Cause	Number	Per cent
Fell down elevator shaft	169	42.2
Crushed between elevator and floor, wall, door, etc	152	37.9
Killed by falling elevator	24	6.0
Struck by elevator	23	5.7
Crushed by counterweights	12	3.0
Crushed in elevator	8	2.0
Struck by falling objects other than elevator	7	1.7
Crushed by machinery of elevator	4	1.0
Unknown how accident occurred	2	.5
Total	401	100.0

TABLE 11.—Summary of Elevator Accidents of Two of the Largest American Cities

[Reported from coroner's records]

	Fatal		Fatal
Borough of Manhattan, New York City, 1907 to 1918, inclusive, public and industrial:	Per cent	Cook County, Ill., 1904 to March, 1916, inclusive:	Per cent
Nonshaft-door accidents	13	Nonshaft-door accidents	19.9
Shaft-door accidents—		Shaft-door accidents—	
Fell down shaft	40.5	Fell down shaft	42.2
Crushed between car and sill or frame of shaft door	46.5	Crushed between elevator and floor, wall, door, etc	37.9
Total	100	Total	100

An analysis of these statistics shows that the weighted average percentage of accidents presented in the preceding tables, due to causes originating from the lack of an interlocking device with which is combined a substantial lock, is as follows:

Fatal accidents to public, per cent of total accidents given in preceding tables	73.8
Fatal accidents to industrial employees, per cent of total	54
Nonfatal accidents to public, per cent of total	40
Nonfatal accidents to industrial employees, per cent of total	53.2

The statistics show that the largest number of accidents is caused by falling down the shaft, the weighted average percentage of the total fatalities to the public from this cause being 36.9, and to industrial employees being 36. In the case of nonfatalities these percentages are, public 21.3, industrial employees 14.7. It should be noted that the percentage of all fatalities from falls is about the same for both classifications, public and industrial. This is a strong indictment of the ordinary door latch, and the results of the survey bear out the conclusion reached by a study of accident statistics. A substantial lock should, therefore, be made a part of every interlock and the ordinary door latch discarded. A somewhat minor point which should not be neglected in the design of interlocks, since it bears directly on the hazard of falling down the shaft, is the possibility of opening the door from the landing side when the car is passing a landing. Devices which permit this should not be installed.

The next most prolific cause of accidents is the crushing hazard; that is, the possibility of being crushed between car and sill of landing floor or frame of door. The average percentage of the total fatalities to the public from this cause is 36.9 and to industrial employees 18. In the case of nonfatalities the percentages are, public 18.7, industrial employees 38.5.

These values, on comparison with those for causes originating within the shaft, show the necessity of interlocking the movement of the car with the opening and locking of the door. Not only should the interlocking function be associated with the closing of the door but also the locking of the door, inasmuch as the unlocked door exposes the user to the hazard of falling into the shaft.

Another cause of accident resulting from lack of interlock devices is stepping into the shaft under the mistaken idea that the car is at the landing. A study of 978 elevator fatalities showed that 1.3 per cent of the accidents were due to this cause, and of 1386 nonfatalities the percentage was 0.9. This is a real hazard and is comparable with many of the accident percentages due to causes originating within the shaft. It is recommended that provision be made against it.

Perhaps the most striking figures are those of the city of Philadelphia, which show the remarkable results obtained by the compulsory use of interlocks in a large city. Of the four people killed on passenger elevators in 1918 only one was a passenger, the other three being elevator operators or mechanics. The fact that there was not a passenger nor an operator killed in an electric elevator during the entire year of 1918 is a remarkable tribute to the effectiveness of the interlocks required by the Philadelphia law and to the administration of the code in force in that city. The contrast in the experience with freight elevators, for which interlocks are not required, is very marked.

The following requirements cover, in general, the desirable interlock features to be incorporated into an elevator code. It is urgently recommended, however, that the latest issue of the A. S. M. E. Elevator Code be consulted as to the exact wording; so that greater uniformity may be secured and that advantage may be taken of criticisms of earlier drafts of that code.

V. SUGGESTED REQUIREMENTS TO BE INCLUDED IN AN ELEVATOR CODE

The following definitions and specifications are submitted as tentative suggestions suitable for inclusion in an elevator code and are intended to give adequate protection and to mitigate the hazards previously set forth. From the evidence thus far accumulated the use of a hoistway-door interlock seems advisable, and this protection should be required on passenger elevators at least.

The material presented in this report was gathered for use in formulating an elevator safety code and was placed at the disposal of the joint committee which was formulating a national code for the American Society of Mechanical Engineers and with which the Bureau of Standards has been cooperating. The definitions and specifications which follow and those tentatively adopted by the committee agree in all essentials. Any evidence tending to support or modify these conclusions will be gladly received by the Bureau of Standards.

1. DEFINITIONS

Hoistway-Door Interlock.—A hoistway-door interlock is a device the purpose of which is—

1. To prevent the normal operation of the car, except by the use of a leveling device, unless (*a*) (door unit system) the hoist-

way door opposite which the car is standing is locked in the closed position; or (*b*) (hoistway unit system) every hoistway door is locked in the closed position.

A hoistway door or gate shall be considered locked in the closed position when within 4 inches of the full closure. If in this position, and any other up to full closure, the door or gate can not be opened from the landing side more than 4 inches.

Interlocks which permit the starting of the car before the door is fully closed shall be so equipped that except when the door is locked in the position of full closure the door or gate can be opened from the landing side to the position approximately 4 inches from full closure.

2. To prevent the opening of a hoistway door (except by use of a key) from the landing side when the car is passing a landing; except when the car-control mechanism is in the "stop" position.

HOISTWAY-DOOR ELECTRIC CONTACT.—A hoistway-door electric contact is an electrical device the purpose of which is to prevent the normal operation of the car except by the use of a leveling device unless (*a*) (door unit system) the hoistway door opposite which the car is standing is in the closed position, or (*b*) (hoistway unit system) every hoistway door is in the closed position.

EMERGENCY RELEASE.—An emergency release is a device the purpose of which is to make hoistway-door electric contacts or hoistway-door interlocks inoperative.

2. SPECIFICATIONS

HOISTWAY-DOOR INTERLOCK SPECIFICATIONS.—(*a*) The prevention of the operation of the car by a hoistway-door interlock shall not be dependent on the action of springs in tension nor solely upon the completion or maintenance of one electrical circuit.

(*b*) The agency used to perform any interlocking function shall be such that even without lubrication of the mechanism the intended functioning of the device will be completely performed.

(*c*) The locking of the hoistway door and the interlocking of the car control shall be accomplished by an interconnection between the parts of the device. This interconnection may be mechanical, electrical, hydraulic, or pneumatic.

(*d*) It shall be necessary to accomplish the locking of the hoistway door opposite which the car is standing before the car can be moved by normal operation.

This paragraph applies to both the door unit and the hoistway unit system.

(*e*) If without damage to, removal of, or interference with any part of the elevator or hoistway equipment the door opposite

which the car is standing becomes unlocked, it shall be impossible to start the car by normal operation.

HOISTWAY-DOOR ELECTRIC CONTACT SPECIFICATIONS.—(*a*) The prevention of the operation of the car by an electric contact shall not be dependent on the action of springs in tension nor solely upon the completion or maintenance of one electric circuit. The failure of the device shall manifest itself by preventing the starting of the elevator from the landing.

(*b*) The device shall be such that without lubrication of the mechanism the intended functioning will be completely performed.

(*c*) All live parts shall be inclosed.

EMERGENCY RELEASE.—(*a*) The emergency release shall be in the elevator car, plainly visible to the occupants and reasonably, but not easily, accessible to the operator.

(*b*) To operate under emergency conditions, it shall be necessary for the operator to hold the emergency release in the emergency position. The emergency release shall be so constructed and installed that it can not be readily tampered with or "plugged" in the emergency position.

(*c*) Rods, connections, and wiring used in the operation of the emergency release that are accessible from the car shall be inclosed to prevent their being tampered with readily.

WASHINGTON, May 14, 1921.

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